Golder Associates Ltd.
CONSULTING ENGINEERS

REPORT TO

CANMET

GEOTECHNICAL STUDY OF ROCK MASS CLASSIFICATION PARAMETERS FOR SYDNEY COALFIELD NOVA SCOTIA

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ABSTRACT

Three rock mass classifications, the Norwegian Geotechnical Institute Q system, the CSIR Geomechanics Classification system and the RSR system were reviewed and parametric and back-analysis were undertaken to establish ranges of values appropriate for use with these classification systems in the coal measures strata of the Sydney Coalfield.

The classification systems were modified to reflect the results of the back-analyses and were then applied to characterize the rock mass at selected existing and future mining areas in the Donkin Mine, Lingan Mine, Prince Mine and Phalen Mine. Correlations between the three classification indices when applied to the coal measures rocks are consistent with those determined by others based on different sets of data for different rock types.

The basis on which back-analyses of the various rock mass classifications were undertaken utilized the support pressure data from Donkin to hold empirical support pressure estimates within close tolerances of envisaged values. Parameter ranges could then be appropriately controlled to ensure compatibility of the classifications with hard rock usage. In this way, the scope of the classifications and the input ratings as modified for the present study have been deliberately cross-checked back to the original intent of each of the ratings. The published support requirement tables therefore remain appropriate and can be used unmodified based on the final classification index values calculated from field observations and applications of the revised classification methods presented in this report.

Some problems remain with accurate predictions of support pressures for all rock mass conditions. The empirical support pressure computation for the CSIR Geomechanics classification appears to give more reasonable results across the entire range of rock types than does the Q-system method. This is because it is based on the rock mass rating as well as the span of the opening. A similar geometrical term needs incorporating into the Q system to take into account arch action as the support pressure estimates for very poor quality rock, with this system far exceed those experienced in practice.

Due to the lack of displacement data and support installation records at the Prince, Lingan and Phalen Mines, further correlations of rock mass classification index data collected from more extensive field work should be undertaken and checked with physically measured and observed performance of the support. Such correlations would aid considerably in projecting the experience gained from this project to other areas of the mines.

SOMMAIRE

Trois systèmes de classification des massifs rocheux; le système Q du Norwegian Geotechnical Institute, le système de classification géomécanique du CSIR et le système RSR, ont été étudiés. Des études paramétriques et des rétro-analyses ont été entreprises afin d'établir des valeurs adéquates pour chacun de ces systèmes en vue de leur utilisation dans les strates carbonifères de la région de Sydney en Nouvelle-Écosse.

Les systèmes de classification des massifs rocheux ont été modifiés pour qu'ils réflètent les résultats des rétro-analyses. Les systèmes ont ensuite été utilisés pour caractériser le massif rocheux dans des chantiers existants (ou en préparation) des mines de charbon Donkin, Lingan, Prince et Phalen. Les trois systèmes de classification des massifs rocheux, lorsqu'appliqués aux roches des strates carbonifères, sont consistants avec les résultats obtenus par d'autres avec des données différentes pour d'autres types de roches.

La base sur laquelle les rétro-analyses des systèmes de classification des massifs rocheux ont été entreprises, faisaient appel aux données sur le chargement du soutènement à la mine Donkin, ceci afin de maintenir les estimés empiriques du chargement du soutènement près des valeurs envisagées. Les marges de variation des paramètres pouvaient alors être contrôlés adéquatement afin de s'assurer de la compatibilité des classifications avec l'usage dans les roches dures.

De cette façon, la portée des systèmes de classification, et les paramètres les composants tels que modifiés pour la présente étude, ont été délibérément contre-vérifiés en les comparant à leurs définitions originales. Ainsi les tableaux déjà publiés indiquant le soutènement nécessaire restent appropriés et peuvent donc être utilisés sans modifications, c'est-à-dire directement à partir des index obtenus à l'aide des systèmes de classification des massifs rocheux et eux-mêmes calculés à partir des observations sur le terrain et des méthodes revisées présentées dans ce rapport.

L'exactitude des prévisions du chargement des soutènements pour toutes les conditions de massifs rocheux demeure toutefois encore problématique. Les calculs empiriques du chargement du soutènement à partir de l'index calculé à l'aide du système de classification géomécanique CSIR semble donner de meilleurs résultats pour tous les types de roches que lorsque calculés à partir de l'index Q. La raison de cette différence est attribuée au fait que l'index du CSIR est basée non seulement sur l'état des massifs rocheux mais aussi sur la grandeur des ouvertures.

Un terme tenant compte de la dimension des ouvertures devrait donc être ajouté au système Q afin de tenir compte de l'effet d'arche puisque les estimés à partir de ce système pour les cas de massifs rocheux de mauvaise qualité excède de beaucoup les chargements mesurés en pratique. En raison du manque de données sur la convergeance et sur l'installation du soutènement aux mines Prince, Lingan et Phalen, plus de travaux sur la corrélation avec les index des systèmes de classification des massifs rocheux tels qu'obtenus à partir de travaux de terrains devraient être entrepris et vérifiés avec des mesures du chargement des soutènements dans ces mines. De telles corrélations aideraient considérablement à étendre l'expérience acquise au cours de ce projet à d'autres secteurs de ces mines.

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1.0 INTRODUCTION

Discussions between the Cape Breton Coal Research Laboratory (CBCRL) and the Cape Breton Development Corporation (CBDC) have revealed considerable interest in the possibility of using rock bolting techniques as a supplementary support system in coal mine roadways and intersections. Despite the potential benefit of using an active support system (such as rock bolts), the change from the traditional support concept which utilizes steel arches to a rock bolting system in the Sydney Coalfield may not be a straightforward transition. It requires an understanding of the geological conditions, a good characterization of the rock behaviour in existing and future mining areas, and a field demonstration of the viability of the proposed support system to the mine operators.

Golder Associates was awarded a research contract under the auspices of CANMET through the Department of Supply and Services (DSS Contract No. 23440-8-9002/01-SQ) to undertake a geotechnical study of rock mass classification methodologies for use in the Sydney Coalfield. Three of the principal rock mass classification systems were specified for review: the Rock Structure Rating (RSR), the Geomechanics Classification (RMR) and the Q-System. The objective of the study was to apply these classification systems to the operating mines in the Sydney Coalfield such that a rational procedure could be developed in the future for determining rock support requirements for the mines.

The scope of the study as completed and reported herein consists of:

a. A brief review and summary of the three rock mass classification systems designated for appraisal,

b. A review of the geological and geotechnical data available from the following mines: Donkin Mine, Lingan Mine, Prince Mine and Phalen Mine,

c. A discussion of the parametric studies and back-analyses undertaken during the study to establish ranges of values appropriate for use with these classification systems in coal measures strata,

d. Several detailed synoptic characterizations of the rock mass at selected existing and future mining areas in the above mines using the three rock mass classification systems and,

e. A comparison of each of the classification systems with reference to their applicability to the Sydney Coalfield.

This report presents the findings of the study and identifies areas where further research efforts are required.

2.0 REVIEW OF ROCK MASS CLASSIFICATION SYSTEMS

2.1 General

Rock mass classification has become an integral component of the empirical rock mechanics design process by providing a quantitative assessment of rock mass conditions. The systematic classification of a rock mass using a standardized system allows those involved in the design process to compare ratings obtained for a particular rock mass with ratings from other engineering projects. Support requirements, stand-up time, and an assessment of maximum unsupported spans can then be evaluated based on previous experience.

Numerous rock mass classification systems have been developed since Terzaghi first introduced his rock load classification (Terzaghi 1946). Table 2.1 summarizes the more common classifications that have been developed and their originators. Lately, most research emphasis has been directed towards modification of existing systems to reflect the experiences gained from the use of these systems for a variety of underground projects (Bieniawski, 1984, CANMET, 1988). In this report, three of the currently applied classification systems are assessed with respect to their usefulness and applicability in classifying the coal measures strata in the Sydney Coalfield. The systems specifically evaluated include the NGI tunnelling quality index (Q) system, the CSIR geomechanics rock mass rating (RMR) system and the rock structure rating (RSR) system. A brief review of each of these systems is presented in the following sections.

2.2 NGI - Q System

The Q system was developed in 1974 by Barton, Lien and Lunde (1974) of the Norwegian Geotechnical Institute (NGI), as a quantitative rock mass classification method. It was based on the analysis of approximately 200 tunnel case histories from Scandinavia, but has since been applied to numerous projects around the world in a wide variety of rock types and geological environments.

The Q system provides a numerical assessment of rock mass quality based on six parameters which are as follows:

- RQD - rock quality designation (Deere, 1964)
- J_n - joint set number
- J_r - joint roughness number
- J_a - joint alteration number
- J_w - joint water reduction number
- SRF - stress reduction factor

The overall rock mass quality, Q, is a function of these six parameters which are combined to form three (3) quotients as follows:

$$Q = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{SRF}$$

The first quotient (RQD/J_n) is intended to be a crude measure of relative block size of material that would fall from the roof of the tunnel or underground opening or would constitute weakness controlled discrete elements within the rock mass fabric. The second quotient (J_r/J_a) is a rough measure of interblock shear strength. The final term (J_w/SRF) involves a water pressure parameter and a 'total stress' parameter, the quotient of which is a measure of in situ 'active stress' conditions. Numerical values for the six parameters for various rock mass conditions are presented in summary in Figure 2.1.

The Q value determined for a rock mass provides a means for estimating support requirements for a given excavation. This is done by first determining the equivalent dimension (D_e) for the underground excavation of concern. "D_e" is a function of both the size and purpose of the opening and is determined as follows:

$$D_e = \frac{S}{ESR}$$

where

D_e is the equivalent dimension of the opening as defined above

S is the span, diameter or height of the opening, and

ESR is the excavation support ratio as defined on Table 2.2.

2.3 CSIR - Geomechanics Classification

The geomechanics classification or rock mass rating (RMR) system provides an alternative quantitative method of assessing rock mass quality and estimating required tunnel support. The RMR system was developed by Bieniawski (1973) of the South African Council for Scientific and Industrial Research (CSIR) in 1973 based largely on experience from tunnel construction. Since its initial development, however, the RMR system has been used in a variety of applications including rock slopes, foundations, underground caverns and hard rock and coal mining (Bieniawski et al 1980).

The RMR classification system is based on assigning a weighted value or rating to the following six parameters:

- uniaxial compressive strength (of an intact specimen of the rock material)

- rock quality designation (RQD)
- spacing of the discontinuities
- condition of the discontinuities
- groundwater conditions
- orientations of the discontinuities

The rating values for each parameter are presented in Figure 2.2. The basic rock mass rating is determined by summing the individual ratings for the first five parameters. The rating for the sixth parameter is a correction factor which is subtracted from the basic RMR value in order to account for the orientation of controlling discontinuities with respect to the underground opening. This correction is applied separately because it is dependent on the engineering purpose of the underground opening.

Based on these weightings, a final factor or adjusted RMR value is derived which gives an indication of rock mass quality. For convenience, the RMR values have been grouped into five classes (see Figure 2.2) ranging from very poor to very good rock.

2.4 Rock Structure Rating

The rock structure rating (RSR) system was developed in 1972 by Wickham, Tiedemann and Skinner for the United States Bureau of Mines. The RSR concept is essentially an empirical approach which also provides a quantitative method of assessing rock mass quality. The results obtained from classifying a rock mass with the RSR system, as with the previously discussed systems, are also intended for use in predicting ground support requirements. For this particular classification, however, the support requirements are heavily biased towards the use of steel rib support methods.

The RSR system was, like the other systems, developed from historical data, literature reviews and ground support theories as well as from considerations relating to the practical aspects of tunnel construction. The data used in establishing the classification system was largely derived from civil engineering projects involving tunnels driven through fairly good ground conditions (Wickham et al, 1972).

The primary factors in the RSR system fall into two general categories: geological parameters and construction parameters. The geological parameters that are considered in the system are: a) rock type, b) joint patterns, c) joint orientation, d) discontinuity conditions, e) structural features (faults, shears and folds), f) groundwater conditions, g) material properties, and h) weathering and alteration. The construction parameters considered include the following: a) size of tunnel, b) direction of drivage, and c) method of excavation.

The RSR rating is then derived from an assessment of the relative effects of each of these factors when combined under three basic classification parameters. These three principal parameter groupings designated as A, B and C are defined as follows:

- Parameter 'A' - General Area Geology
  - describes the basic rock type, rock hardness and geologic structure
- Parameter 'B' - Joint Pattern - Direction of Drivage
  - describes the joint spacing, joint orientation and direction of tunnel drivage

- Parameter 'C' - Groundwater - Joint Condition
  - describes the rock mass quality determined from parameters A and B with the joint condition and water inflow

Ranges of values for these three parameter groupings are summarized in Figure 2.3. To determine the final RSR value, the weighted numerical values obtained for parameters A, B and C are summed. The final rating so computed has a range from 19 to 100 depending on rock quality with higher numbers indicating better ground conditions.

An additional adjustment to the basic RSR value to further account for the effects of the excavation method on rock mass conditions has also been suggested by Wickham, et al (1972) specifically to adjust the rating for machine bored tunnels as a function of tunnel diameter (see Figure 2.4).

TABLE 2.1

COMMONLY USED ROCK MASS CLASSIFICATION SYSTEMS

Classification System	Originator and Date	Country of Origin	Applications
Rock loads	Terzaghi, 1946	USA	Tunnels with steel supports
Stand-up time	Lauffer, 1958	Austria	Tunnelling
Rock quality designation	Deere, 1964	USA	Core logging, tunnelling
Intact rock strength	Deere & Miller 1966	USA	Communication Tunnels
RSR concept	Wickham, et al. 1972	USA	Tunnelling
Geomechanics Classification (RMR system)	Bieniawski, 1973	S. Africa & USA	Tunnels, mines foundations
Q-system	Barton, et al. 1974	Norway	Tunnelling, Mining, large chambers
Strength/block size	Franklin, 1975	Canada	Tunnelling
Basic geotechnical classification	ISRM, 1981	International	General

Adapted from Bieniawski (1984).

TABLE 2.2

EXCAVATION SUPPORT RATIO VALUES

NGI-Q SYSTEM

	Excavation Category	ESR	No. of Cases
A.	Temporary mine openings	3-5	2
B.	Vertical shafts:		
	circular section	2.5	--
	rectangular/square section	2.0	--
C.	Permanent mine openings, water tunnels for hydropower (excluding high-pressure penstocks), pilot tunnels, drifts and headings for large excavations.	1.6	83
D.	Storage rooms, water treatment plants, minor highway and railroad tunnels, surge chambers, access tunnels.	1.3	25
E.	Power stations, major highway or railroad tunnels, civil defense chambers, portals, intersections.	1.0	73
F.	Underground nuclear power stations, railroad stations, factories.	0.8	2

Adapted from Barton et al. (1974).

NGI ROCK CLASSIFICATION SYSTEM PARAMETER EVALUATION

FIGURE 2.1

Item and Description	Parameter	Value
Number of Sets of Discontinuities	J_n	
Massive		0.5
One set		2.0
Two sets		4.0
Three sets		9.0
Four or more sets		15.0
Crushed rock		20.0
Roughness of Discontinuities	J_r	
Noncontinuous joints		4.0*
Rough and wavy		3.0*
Smooth and wavy		2.0*
Rough and planar		1.5*
Smooth and planar		1.0*
Slick and planar		0.5*
Filled discontinuities		1.0*
Filling and Wall-Rock Alteration, Essentially Unfilled	J_a	
Healed		0.75
Staining only; no alteration		1.0
Silty or sandy coatings		3.0
Clay coatings		4.0
Filling and Wall-Rock Alteration, Filled	J_a	
Sand or crushed rock filling		4.0
Stiff clay filling less than 5 mm thick		6.0
Soft clay filling less than 5 mm thick		8.0
Swelling clay filling less than 5 mm thick		12.0
Stiff clay filling more than 5 mm thick		10.0
Soft clay filling more than 5 mm thick		15.0
Swelling clay filling more than 5 mm thick		20.0
Water Conditions	J_w	
Dry		1.0
Medium water inflow		0.66
Large inflow; unfilled joints		0.5
Large inflow; filled joints with washout		0.33
Large inflow; filled joints; high transient inflow		0.2 to 0.1
Large inflow; filled joints; high continuous inflow		0.1 to 0.05
Stress Reduction Class	SRF	
Loose rock with clay-filled discontinuities		10.0
Loose rock with open discontinuities		5.0
Shallow depth (50 m or less) rock with clay-filled discontinuities		2.5
Rock with tight unfilled discontinuities; medium stress		1.0
Rock-Mass Quality for Tunneling	Q	
Exceptionally poor		Less than 0.01
Extremely poor		0.01 to 0.10
Very poor		0.10 to 1.00
Poor		1.00 to 4.00
Fair		4.00 to 10.0
Good		10.0 to 40.0
Very good		40.0 to 100.0
Extremely good		100.0 to 400.0
Exceptionally good		400.0 to 1000.0

Adapted from: Barton, N., Lien, R., and Lunde, J., 1974.
* Add 1.0 if the mean joint spacing exceeds 3.0 m.

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CSIR GEOMECHANICS CLASSIFICATION FOR JOINTED ROCK MASSES

FIGURE 2.2

A. Classification parameters and their ratings

PARAMETER			RANGES OF VALUES						
1	Strength of intact rock material	Point-load strength index	> 10 MPa	4 - 10 MPa	2 - 4 MPa	1 - 2 MPa	For this low range – uniaxial compressive test is preferred		
		Uniaxial compressive strength	>250 MPa	100 - 250 MPa	50 - 100 MPa	25 - 50 MPa	5-25 MPa	1-5 MPa	< 1 MPa
	Rating		15	12	7	4	2	1	0
2	Drill core quality RQD		90% - 100%	75% - 90%	50% - 75%	25% - 50%	< 25%		
	Rating		20	17	13	8	3		
3	Spacing of discontinuities		>2 m	0.6 - 2 m	200 - 600 mm	60 -200 mm	60 mm		
	Rating		20	15	10	8	5		
4	Condition of discontinuities		Very rough surfaces Not continuous No seperation Unweathered wall rock	Slightly rough surfaces. Separation < 1 mm Slightly weathered walls	Slightly rough surfaces Seperation < 1 mm Highly weathered walls	Slickensided surfaces OR Gouge < 5 mm thick OR Separation 1-5 mm Continuous	Soft gouge > 5mm thick OR Separation > 5 mm Continous		
	Rating		30	25	20	10	0		
5	Ground water	Inflow per 10 m tunnel length	None	<10 litres/min	10-25 litres/min	25 - 125 litres/min	> 125		
		Ratio $\frac{\text{joint water pressure}}{\text{major principal stress}}$	OR 0	OR 0.0-0.1	OR 0.1-0.2	OR 0.2-0.5	OR > 0.5		
		General conditions	OR Completely dry	OR Damp	OR Wet	OR Dripping	OR Flowing		
	Rating		15	10	7	4	0		

B. Rating adjustment for discontinuity orientations

Strike and dip orientations of joints		Very favourable	Favourable	Fair	Unfavourable	Very unfavourable
Ratings	Tunnels	0	-2	-5	-10	-12
	Foundations	0	-2	-7	-15	-25
	Slopes	0	-5	-25	-50	-60

C. Rock mass classes determined from total ratings

Rating	100←81	80←61	60←41	40←21	< 20
Class No	I	II	III	IV	V
Description	Very good rock	Good rock	Fair rock	Poor rock	Very poor rock

D. Meaning of rock mass classes

Class No	I	II	III	IV	V
Average stand-up time	10 years for 15 m span	6 months for 8 m span	1 week for 5 m span	10 hours for 2.5 m span	30 minutes for 1 m span
Cohesion of the rock mass	> 400 kPa	300 - 400 kPa	200 - 300 kPa	100 - 200 kPa	< 100 kPa
Friction angle of the rock mass	< 45°	35° - 45°	25° - 35°	15° - 25°	< 15°

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RSR ROCK CLASSIFICATION SYSTEM PARAMETER EVALUATION

FIGURE 2.3

Rock structure rating – Parameter *A*: general area geology (after Wickham et al., 1974)

	Basic rock type				Geological structure			
	Hard	Med.	Soft	Decomp.	Massive	Slightly faulted or folded	Moderately faulted or folded	Intensely faulted or folded
Igneous	1	2	3	4				
Metamorphic	1	2	3	4				
Sedimentary	2	3	4	4				
Type 1					30	22	15	9
Type 2					27	20	13	8
Type 3					24	18	12	7
Type 4					19	15	10	6

Rock structure rating – Parameter *B*: joint pattern, direction of drive (after Wickham et al., 1974)

Average joint spacing	Strike ⊥ to axis					Strike ∥ to axis		
	Direction of drive					Direction of drive		
	Both	With dip		Against dip		Both		
	Dip of prominent joints*					Dip of prominent joints*		
	Flat	Dipping	Vertical	Dipping	Vertical	Flat	Dipping	Vertical
1. Very closely jointed < 2 in.	9	11	13	10	12	9	9	7
2. Closely jointed 2–6 in.	13	16	19	15	17	14	14	11
3. Moderately jointed 6–12 in.	23	24	28	19	22	23	23	19
4. Moderate to blocky 1–2 ft.	30	32	36	25	28	30	28	24
5. Blocky to massive 2–4 ft.	36	38	40	33	35	36	34	28
6. Massive > 4 ft.	40	43	45	37	40	40	38	34

Rock structure rating – Parameter *C*: ground water, joint condition (after Wickham et al., 1974)

Anticipated water inflow (gpm/1000 ft)	Sum of parameters *A* + *B*					
	13–44			45–75		
	Joint condition**					
	Good	Fair	Poor	Good	Fair	Poor
None	22	18	12	25	22	18
Slight < 200 gpm	19	15	9	23	19	14
Moderate 200–1000 gpm	15	11	7	21	16	12
Heavy > 1000 gpm	10	8	6	18	14	10

*Dip: flat: 0–20 deg; dipping: 20–50 deg; and vertical: 50–90 deg.

**Joint condition: Good = tight or cemented; Fair = slightly weathered or altered; Poor = severely weathered, altered, or open.

ADAPTED FROM WICKHAM et al (1972)

Date MARCH, 1989

Project 881-1719

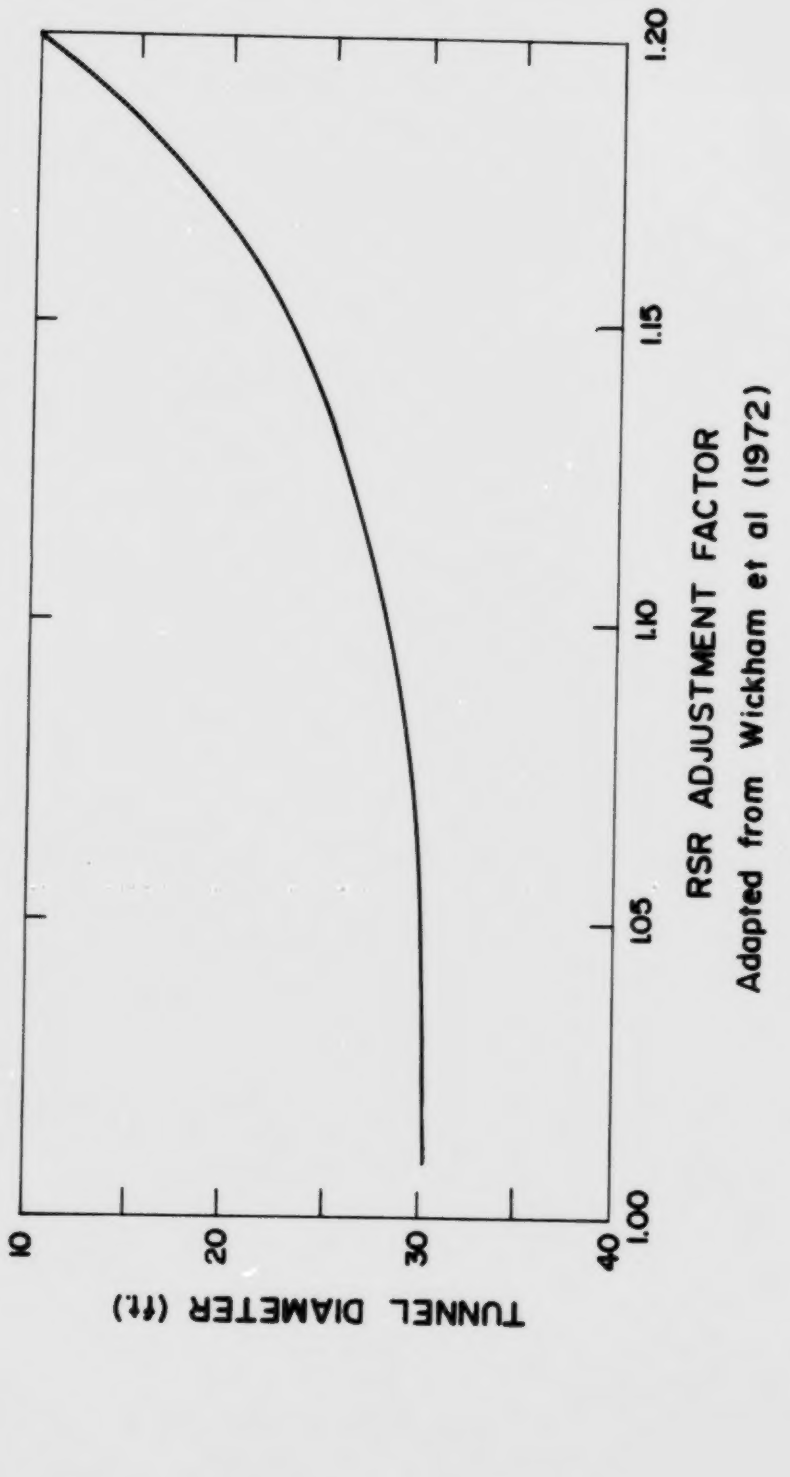
Golder Associates

Drawn CFM

Chkd

RSR ADJUSTMENT FOR MACHINE BORED TUNNELS

FIGURE 2.4



Adapted from Wickham et al (1972)

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Drawn CPAM
Chkd

Form G.A-U-4m (metric)

3.0 GEOLOGICAL AND MINING CONDITIONS OF THE SYDNEY COALFIELD

3.1 Regional Geology

The Sydney Coalfield (Figure 3.1) contains the largest coal reserve in eastern Canada. The productive coal measures lie within a 2200 m sequence of strata known as the Pictou Group. The most extensive and youngest coal deposits of Nova Scotia occur within this group. Eleven major coal seams (six of them are of economic importance) are present, ranging in thickness between 1.0 to 4.5 m. The coals are late Pennsylvanian in age and are ranked as high volatile "A" bituminous coals (Forgeron et al 1986).

The strata between the coal seams consist of sandstone, siltstone, mudstone, minor shale and limestone occurring in relatively consistent intervals. The entire sequence bears the characteristics of deposition in a fluvial environment upon a slowly subsiding coastal plain.

The coalfield has not been appreciably disturbed tectonically. Major structures in the vicinity are the bounding faults along the margin of the coal basin (the Mountain Fault to the west and the Mira River - Bateston fault to the south east), and the large-scale north-east trending folds which define a broad structure known as the Sydney synclinorium. In general, in the area of current mining, the coal seams dip north at 4 to 10 degrees, while the gradient on the flanks of the folds towards the bounding faults rarely exceeds 15 degrees.

3.2 Effects of Geological Features on Coal Mining in the Sydney Coalfield

All the underground mining operations in the Sydney Coalfield are owned and operated by the Cape Breton Development Corporation (CBDC). The operating coal mines include the Prince Mine at Point Aconi (on the Hub Seam), the Lingan Mine at New Waterford (on the Harbour seam) and the Phalen mine at New Waterford (on the Phalen seam) adjacent to the Lingan Mine. The No. 26 Colliery at Glace Bay (on the Harbour seam) was closed in April 1984 due to an uncontrollable fire in the mine. The Donkin Mine at Donkin-Morien (on the Harbour seam) is currently on hold after completion of the two mine access tunnels driven to intersect the Harbour seam at approximately 3.5 km offshore.

The Geological Department of CBDC has maintained an extensive record of geological mapping along some 150 km of underground workings and has compiled more than 200 case records of geologically-related mining problems in the operating coal mines (Forgeron et al 1986).

On the basis of this data, Forgeron has summarized the impact of geological features on the safe and efficient extraction of coal in the Sydney Coalfield. Specifically, he has identified six main points as follows:

a. Although the operating mines are generally dry, water (usually a minor amount) is present as a result of seepage along intergranular boundaries and geological weakness (lithological contacts, fossil partings, joints and tension fractures, etc.) or introduced during the mining process. The presence of water generally tends to reduce the durability and self-supporting strength of both the roof and floor strata.

b. Contacts between lithological units of markedly different composition tend to be weak. Such contacts include shale-coal, sandstone-coal, sandstone-shale. Where these contacts occur within close proximity of the roof (approximate depth equal to half of the span) the potential for unstable roof conditions is greatly increased.

c. Strata that are thinly laminated and have an abundance of plant debris parallel to the bedding planes, tend to promote bed separation in the roof, thus resulting in a deeper loosened zone at the roof.

d. Sandstone intrusions (paleo-river channels) replacing the normal roof sequence of the coal are observed in the mines. Problems usually occur when the sandstone intrusions occur within the seam or in close proximity (generally less than 2 m) of the roof of the mine openings. Some of the specifically observed effects of the sandstone channels on the mining operation have also been summarized (Forgeron, 1986) as follows:

   - the rock immediately adjacent to rapid sandstone pinch-out zones tends to be weak. Rock falls as high as 2 m have been reported where such units occur in the roof.

   - the sandstone-mudstone contacts always constitute planes of weakness along which rock immediately separates when coal is removed.

   - the channels reduce or completely destroy the natural beaming strength of the roof strata.

   - outbursts generally originate from such sandstone overbank deposits above the seam below a depth of

700 m. Five of these sandstone channels were identified in No. 26 Colliery and outbursts were recorded where the sandstone channel came within 3 m of the top of the coal seam and during shotfiring.

- the sandstone channels are limited in width and their geometric shapes are difficult to delineate in three dimensions using borehole data.

e. All the productive coal seams in the Sydney Coalfield exhibit stratigraphic splitting probably reflecting interruptions during the deposition of the original peat bogs. The effect of such splits is to produce stone (non-coal) bands within a major coal seam. On occasion these divide the seam into one or more subseams, some of which may not be individually mineable. Unstable roof conditions have been experienced in these zones primarily due to delamination at the coal/stone contact surfaces.

f. Irregular masses of silty mudstone with little identifiable internal structure have been encountered in the room and pillar section at the Lingan Colliery. These are inferred to represent infilled rupture features created during the period of original peat deposition. Where these features extend into the overlying strata, destabilization of the immediate roof may occur predominantly due to the irregular nature of such mudstone units in the roof. This leads to a reduction in the intrinsic strength of the normal roof strata (generally siltstone or sandstone) and thus can promote instability. Similar features but consisting of siltstones and fine sandstones which exhibit cross-bedding characteristics, have also on occasion given rise to similar instability problems.

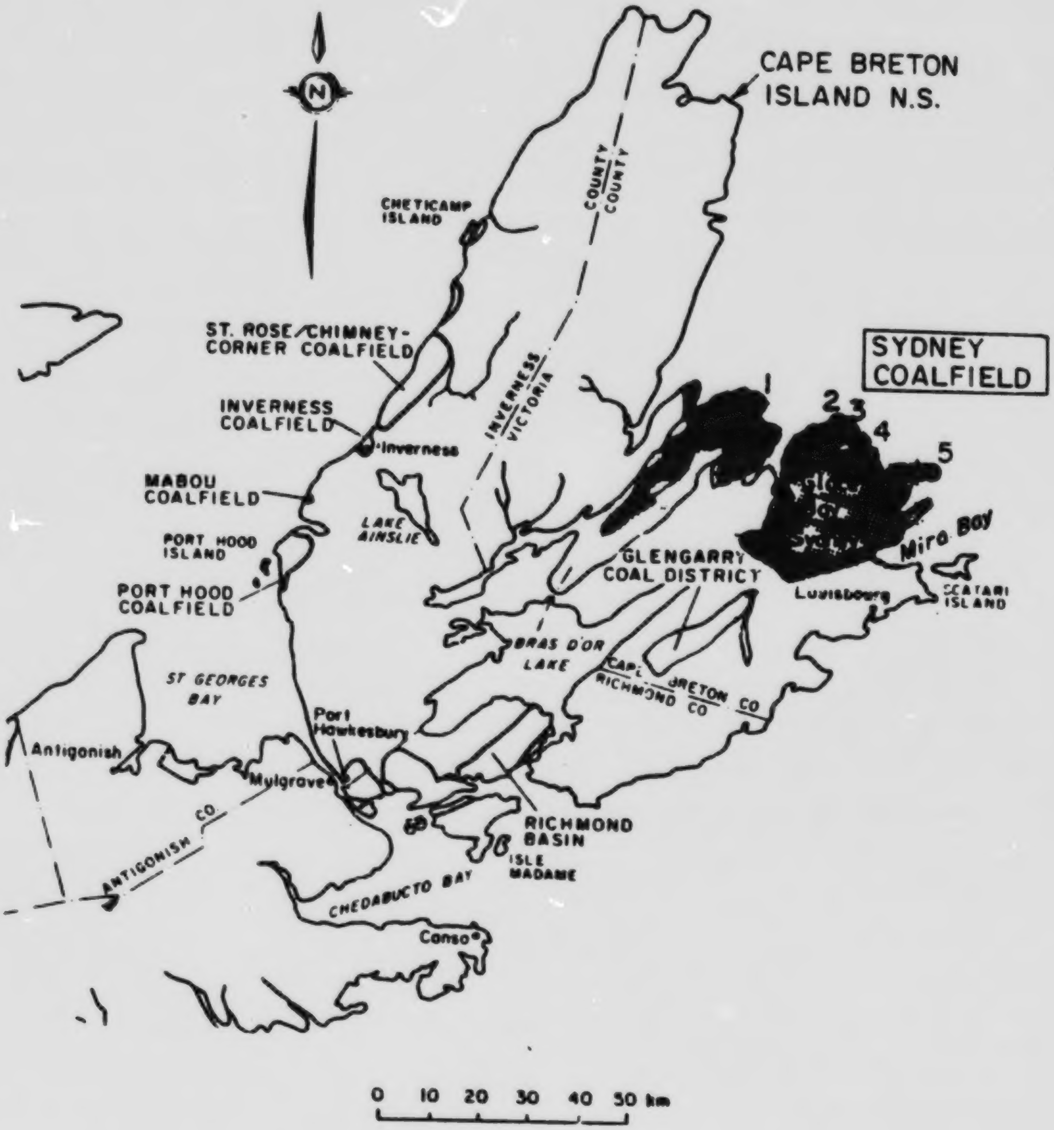
g. Seat-earths encountered in the Sydney Coalfield are highly variable in composition. They range from mudstones to quartzose sandstones and usually occur in the immediate floor. In such situations floor conditions can deteriorate markedly from hard working surfaces where sandstones are present to soft surfaces where seat-earths occur. The bearing capacity of the seat-earths are so low that they may be penetrated by the legs of the roof support structure.

Although the general descriptions of the geological features and their effects on the stability of mine openings presented above are intended to provide an overview of the influence of geological factors on the mining environment of the Sydney Coalfield, they also provide some of the basic data utilized for developing appropriate classification approaches for using the various rock mass characterization systems.

The behaviour of the rock mass to mining, however, is very site specific, depending not only on rock mass strength and structure but also on excavation spans and on the support installation procedure which significantly affects the effectiveness of the support system. The site specific characteristics of the coal measures strata encountered in each of the mines in the coalfield and their different behaviour also must be considered when determining appropriate rock classification parameters. These points are reviewed in more detail in the following sections of this report.

LOCATION MAP OF SYDNEY COALFIELD, NOVA SCOTIA

FIGURE 3.1


N
CAPE BRETON ISLAND N.S.
CHETICAMP ISLAND
COUNTY
COUNTY
INVERNESS
VICTORIA
ST. ROSE/CHIMNEY-CORNER COALFIELD
INVERNESS COALFIELD
Inverness
MABOU COALFIELD
LAKE AINSLIE
PORT HOOD ISLAND
PORT HOOD COALFIELD
SYDNEY COALFIELD
1
2
3
4
5
Mira Bay
GLENGARRY COAL DISTRICT
Louisbourg
SCATARI ISLAND
BRAS D'OR LAKE
CAPE BRETON CO
RICHMOND CO
ST GEORGES BAY
Port Hawkesbury
Antigonish
Mulgrave
ANTIGONISH CO
RICHMOND BASIN
ISLE MADAME
CHEDABUCTO BAY
Canso
0 10 20 30 40 50 km

UNDERGROUND MINES IN THE SYDNEY COALFIELD

1. PRINCE MINE
2. LINGAN MINE
3. PHALEN MINE
4. No. 26 COLLIERY (CURRENTLY CLOSED)
5. DONKIN-MORIEN MINE (DEVELOPMENT ON HOLD)

LEGEND

LCM LCP LCS — PICTOU, STELLARTON AND MORIEN GROUPS (STRATIGRAPHIC GROUP OF SYDNEY COALFIELD)

Date MARCH, 1989
Project 881-1719

Golder Associates

Drawn CPAM
Chkd

4.0 APPLICATION OF ROCK MASS CLASSIFICATION SYSTEMS FOR USE IN COAL MEASURES ROCKS

4.1 Background

The application of rock mass classification schemes to coal mining has not been as extensive as in civil engineering and other mining operations. Steel sets have traditionally been used for rock support in coal mines in various countries (e.g. U.K., Europe, Canada). In the United States and Australia, rock bolting is extensively used for both room and pillar and longwall coal mining operations. While there is a shifting trend towards the use of rock bolts as an alternative active support system for both longwall and main roadways, precedent practices are still largely followed in most coal production areas.

A review of the support design procedures adopted by various coal mining operations in different countries (CANMET, 1988) indicated that steel sets were generally designed on the basis of some assumed rock load distribution based on the local mining experience in that particular area. The use of rock mass classifications for support design has, to date, been very limited.

On the basis of published literature, the only systematic use of rock mass classification methods for developing support design for coal mines has been undertaken in the United States for room and pillar operations (Bieniawski, 1980, Unal, 1983). In these cases the CSIR rock mass rating has been used as an indirect means of deriving the anticipated depth of loosened rock above the mine openings according to the following empirical expression (Unal, 1983):

$$h_e = \frac{(100-RMR)}{100} \times B \tag{4.1}$$

where h_e = Depth of loosening zone above the mine opening

RMR = CSIR rock mass rating of the roof strata

B = width of the opening

This expression was specifically developed to aid in coal mine roof support design. It was based on a review of a series of rock falls and of overall performance of various mine openings in a number of operating coal mines in the United States. The CSIR (RMR) ratings appropriate to the roof strata in the areas of the mines reviewed were determined on the basis of the conventional classification procedures proposed by Bieniawski (1973). Specific consideration was given, however, to the layering of the roof strata by weighting the RMR ratings according to the thickness of the individual layers within the immediate roof areas (approximately up to a height equal to h_e). Support recommendations were then developed based on the use of mechanical and resin grouted rock bolts of appropriate lengths and strength to match the height of loosened rock for various spans and rock mass ratings for the typical rectangular openings and intersections evaluated with the CSIR system.

4.2 Generic Application of Rock Mass Classifications to the Sydney Coalfield Rocks

Based on the geological conditions in the Sydney Coalfield and the observed performance of the mine openings in the operating coal mines (Forgeron 1986), it has been recognized that in many cases roof failures in coal measures strata tend to occur by delamination at the contact of different rock types and by tensile splitting of discrete subhorizontal bedding surfaces within individual units. At the gate roads located adjacent to the advancing longwall panels, failures are further influenced by the induced fracturing of the roof strata due to excessive deformation of the rock units above the mined out longwall panels. Hence, the characteristics of the bedding surfaces and the strength of the predominantly sub-horizontal discontinuities (including bedding planes and contacts between rock types) have a significant effect on the overall rock mass quality which in turn can be related to the overall stability state of the underground openings.

Unfortunately, the current rock mass classification schemes, including the three systems under consideration in this study, do not directly allow characterization of this most important weakness characteristic inherent in sedimentary rock sequences. For example, in the NGI-Q system, as currently defined, it is difficult to relate the J_n parameter, a measure of jointing intensity, to the bedded nature of coal measures strata. Similarly, J_a values for surface condition of joints were based on hard rock environments and do not directly relate to the characteristics of weaknesses within sedimentary rock strata.

In the CSIR-RMR system, the direct application of the rating for spacing of discontinuities also requires a considerable amount of judgement on the user's part because such ratings were originally derived for hard rock conditions. Furthermore, the ratings for discontinuity conditions and the adjustment for strike and dip directions are also not directly applicable to coal measures rocks.

The geological input parameters necessary to utilize the RSR system are, because they are so general, even more difficult to apply than the Q and RMR systems; thus again their direct application to coal measures rocks also requires further consideration.

Hence, in order to apply these three rock mass classification systems to the Sydney Coalfield in a consistent and rational manner, the fundamental concepts of the classification systems have had to be re-examined and a rationale for selecting appropriate values for the individual parameters developed for each system in order to match known roof conditions and behaviour.

4.3 Back-Analysis of Classification Indices

Considerable discrepancy becomes apparent when one tries to apply any of the existing classification systems to coal measures rocks. The approach adopted in this study has therefore been to utilize available information on rock support performance to establish viable parameter descriptors for each classification component. This has been accomplished by undertaking quantitative back-analysis of actual rock loads that have been acting onto installed support systems as estimated on the basis of monitoring records and evaluating these inferred loads against the observed performance of the support system.

Among the four mines under consideration in this study, data from the tunnelling aspects of the main drivages for the Donkin Mine provides the most comprehensive cross-section of rock types of any of the coal mines. This project comprised the excavation of two parallel 7.6 m diameter decline tunnels (Donkin Tunnel No. 2 and 3) through the coal measures strata to intersect the Harbour coal seam at about 3.5 km offshore. An extensive geotechnical monitoring program (CANMET 1985) consisting of rock deformation measurements and support performance evaluation was undertaken during the tunnel drivage. Good correlation information was available relating rock mass conditions and rock types to support performance. Thus this data base has been used to quantitatively relate the performance of the rock mass to underground excavation development.

Of the data collected as part of the 1985 monitoring program, the records of the extensometers installed at the roof of the tunnel are particularly useful as a means for delineating the zone of active displacement above the crown of the tunnel. As much of the Donkin Tunnels were driven by a TBM and the tunnel supports (consisting of a series of 150 mm x 150 mm circular ring beams at 1.5 m cc) were installed within 8 m of the face and shortly after excavation, rock loosening was kept to a minimum. The zone of active displacement recorded by the extensometers, therefore, can be considered to fairly reliably represent a measure of the rock load which would reasonably be expected from the rock mass onto a properly supported tunnel under that particular drivage condition. The support pressure which could be inferred from the extensometer data is considered to reflect, as closely as practicable, true rock mass response data without the complicating effects of blast-induced loosening and inconsistent support procedures. It has thus been utilized as the primary basis from which to develop appropriate classification descriptors for generic application of rock classification methods to other locations in the coal mining environment of the Sydney Coalfield.

4.4 Development of Appropriate Classification Descriptors

Among the three classification systems under consideration, the Q system provides the most flexible method for tackling the problem of truly characterizing rock behaviour. As originally conceived, the Q system was intended to provide an empirical approach to categorize variations in rock mass shear strength behaviour along given controlling discontinuities within the rock mass. Expressed in terms of the Mohr-Coulomb shear stress relationship, this is given by equation (4.2) as:

$$\tau = \sigma_n' \tan \phi' \qquad (4.2)$$

where σ_n' is the effective normal stress acting on the discontinuity surface and angle ϕ' is the frictional angle of the discontinuity surface

The Q index when expressed in the manner shown in equation (4.3) can also be thought of as having the same general formulation as equation (4.2).

$$Q = \frac{RQD}{J_n} \times \frac{J_w}{SRF} \times \frac{J_r}{J_a} \qquad (4.3)$$

where RQD/J_n = normal stress/weight/block size term
J_r/J_a = frictional strength term
J_w/SRF = effective stress/water pressure term

That is, it effectively constitutes a crude measure of the shear resistance of the rock mass. Thus, by careful evaluation of actual rock mass behaviour, it becomes possible to estimate Q values that are appropriate for given conditions and hence by inference back-calculate optimum sub-parameter input values for the Q system.

Numerous research workers have undertaken empirical correlations both between the various other classification systems and also with respect to required/necessary support pressures Figure 4.1 illustrates two published support pressure/classification index correlation graphs and Figure 4.2 presents one of the many published graphs relating the Q index to the CSIR (RMR) ratings. These plots form the backbone of the proposed classification approach outlined in the following paragraphs.

In order to relate Q values and support pressures to actual rock conditions, the data from each of the ten available instrumented locations along the TBM drivage (in Tunnel No. 2) at Donkin were carefully examined. This detailed appraisal of each location has allowed relationships to be developed between support pressures and rock mass characteristics. This then has formed the basis for deriving a range of values and descriptors for all the rock classification parameters needed to evaluate Q or RMR. The terms relating to the Q system were derived first followed by the terms for the CSIR (RMR) system and finally, the RSR descriptor units were derived by cross-correlation.

4.4.1 Parameters for the NGI-Q Systems

As an illustration of the methodology adopted for this back-analysis, the records from the extensometer installation at Ch. 1889 m in the TBM drivage at Donkin are discussed in detail. The approach taken for deriving the various descriptor ratings presented on Table 4.1 for this and other locations is illustrated by reference to the flow chart shown on Figure 4.3.

The rock types encountered at the tunnel roof at Ch. 1889 m are summarized as follows:

Rock Type	Thickness (m)	Distance from Crown of Tunnel (m)
Interbedded Siltstone and Mudstone	2.2	
--------------------------------------		3.5
Thinly Bedded Interbedded Sandstone and Siltstone	3.5	
---------- roof ----------------------		0

The individual parameters needed for deriving the NGI rock quality index, Q, are discussed in the following sections using the above location as reference.

RQD Parameter

To provide a measure of the first parameter descriptor for the Q system, some measure of the corability of the rock is required. Initially, assessments of appropriate RQD values, as well as base friction angles and average block sizes (interpreted as slab thickness) were made based on the data from the extensometer records and from face and sidewall mapping information.

As only some of the extensometer holes had been cored, the information that could be used directly for RQD determinations was limited. Reliance was therefore placed on the detailed geological descriptions provided from the face and sidewall mapping records. Use was made of the Deere (1966) and Priest and Hudson (1981) graphs (Figures 4.4 and 4.5 respectively) to indirectly derive RQD values from these diagrams direct correlation between fracture frequency and/or mean discontinuity spacing and RQD is possible.

Using this approach, an RQD of 60% was assigned for Ch. 1889 based on Figures 4.4 and 4.5 assuming a mean discontinuity spacing of about 100 mm or a fracture frequency of 3 fractures per 1 foot for the thinly bedded sandstone/siltstone at the roof line of the tunnel.

J_n Parameter

In order to allow assessment of the J_n parameter, some idea is needed of the characteristic block size or slab thickness that might fall from the tunnel crown. For actual field conditions, this is usually fairly easy to assess, but is somewhat more problematic to define solely on core alone. Nevertheless, as an initial estimate, the controlling bedding plane spacing can be used as a crude measure of slab thickness. For the Ch. 1889 situation based on appraisal of both rock behaviour during tunnelling and fracture spacing in the core the following value of block size was selected for the controlling bedding plane weaknesses:

Average Block Size = 5 cm

Using these field assessed parameters, an appropriate value for J_n was then back-calculated from the relationship:

$$J_n = \frac{\text{RQD (in percent)}}{\text{Block Size (in cm)}} \tag{4.4}$$

$$= \frac{60}{5}$$

$$= 12$$

The J_n parameter in the original Q-system proposed by Barton et al (1974) was defined as a measure of the number of controlling joint sets existing within the rock mass that governed the size of blocks bounded by such joints. In horizontally bedded sedimentary rock, the descriptors given in the standard tables (Figures 2.1 and 2.2) are difficult if not impossible to rationally use to define J_n values appropriate to the rock mass. These descriptors and hence the J_n parameter ratings have therefore been modified to reflect the importance of bed thickness rather than joint sets in determining block size. Values for this modified J_n, which have been kept within the same range from 0.5 to 30 for massive to completely fissured rock respectively, are shown on Table 4.2. Although the descriptors are different from Barton's originals, they are still intrinsically comparable to the original values and concepts of the Q system (compare Figure 2.1 and Table 4.2).

J_r Parameter

The roughness parameter, J_r, as originally defined by Barton (1974) provides a measure which reflects variations in surface characteristics of controlling discontinuities (Figure 2.1). As the descriptive codes characterize surface conditions of discontinuities and not the type of discontinuities, the standard Q system descriptors for joint surfaces are still considered relevant and thus can be applied directly for assessing bedding plane conditions within horizontally bedded strata without modifications to the original index values. However, in order to provide better definition of the smoother surfaces, several additional descriptor terms have been added. The rating value range has been kept consistent from 0.5 to 4.0 for slickensided, planar surfaces to non-continuous discontinuities respectively (see Table 4.2).

J_a Parameter

The J_a parameter, reflects the surface condition of the rock material within which the discontinuities exist and their alteration and/or infilling. Based on the original tabulations by Barton (1974) this parameter J_a can be roughly related to the residual friction angle assessed for smooth surfaces (such as sawn surfaces for the material considered representative of the discontinuity walls/surfaces. Based on the original published ranges and using data from other available records as a guide, a new set of descriptor terminology and assigned approximate residual friction angles have been developed for typical coal measure rocks (Table 4.2). Again, as with the J_n and

J_r parameters, the range of descriptor codes and parameter values have been kept as close as possible to the original intent of the classification.

Values of J_a and J_r are intimately related according to the original definition of the parameters, namely, the quotient J_a/J_r is defined as being roughly equivalent to $\tan^{-1}(\phi')$. Further, a direct relationship between J_a and the residual friction angle ϕ_r can be derived using the original published data (Barton 1974) and the wider range of values presented in Table 4.2. This is graphically shown in the top diagram on Figure 4.6. The lower diagram extends this relationship to include J_r, allowing the peak shear strength for a given discontinuity condition to be computed knowing that:

$$\phi'_{peak} = \tan^{-1}\left(\frac{J_r}{6000\ \phi_r^{-2.4}}\right)$$

These graphs can then be rapidly used to cross check the parameter descriptors and verify the selected parameters. For the situation at ch. 1889, a residual shear strength of 220° is estimated for the interbedded sandstone/siltstone at the site. A J_a value of 3.5 based on the top graph on Figure 4.6 seems consistent with the descriptor codes for the contact surface (Ref. Table 4.2). Actually, these descriptor codes for the J_a parameter (Table 4.2) were iteratively developed from the back-analysis to ensure a match to measured pressures (Table 4.1). However, as an illustration of their application, once a J_a value has been decided, a J_r value can be selected from Table 4.2. For ch. 1889 a value of 2.0 is originally estimated for the type of bedding planes at the site giving an estimated peak friction angle of:

$$\phi'_{peak} = \tan^{-1}(J_r/J_a) = \tan^{-1}(2.0/3.5) = 28^{\circ}$$

with several iterations, with all the relationships set up on a spreadsheet program, a balanced set of parameters and parameter values can be obtained. As shown in Table 4.1, the actual best match values for ch. 1889 are J_a=3.6, J_r=1.9 respectively.

On the basis of this evaluation presented above and on the others undertaken for the range of conditions at the Donkin Tunnels, the values and descriptors for J_a, the joint alteration number, as originally termed (Figure 2.1), have been carefully modified to accurately describe and quantify the surface conditions of the range of characteristic bedding plane contacts as these form the most critical discontinuities in sedimentary rocks (see Table 4.2). However, in order to maintain consistent J_a/J_r ratings with those for similar strength hard rock discontinuity conditions, the range of ratings for J_a in particular have been also cross-checked back to the original intent and tabulations of the Q system.

J_w/SRF Parameter

The J_w/SRF quotient, which is a crude measure of active stress and groundwater influence, is considered to be independent of rock type. Thus the descriptors for these parameters have not been modified from the original Q system tabulations (Figure 2.1) although as discussed later, some suggestions are made for SRF values that can reflect disturbance effects due to longwall proximity. For the Donkin case, specifically, Ch. 1889, therefore, the groundwater and SRF parameters were assigned values of 1.0 for dry conditions and 1.0 for medium stress, respectively (refer to Table 4.1).

Q Index

The final Q rating for this Ch. 1889, example was then calculated according to Barton's original Q system equation as follows:

$$Q = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{SRF}$$

$$= \frac{60}{12} \times \frac{1.9}{3.6} \times \frac{1.0}{1.0}$$

$$= 2.7$$

Support Pressure Correlation

As the basis for the back-analysis assessment, the Q index value calculated from the above approach was checked to see if it would predict support pressures consistent with the appropriate support needed for the observed underground conditions. Correlation calculations were carried out to evaluate the support pressures predicted by the empirical relationship presented by Barton (1973) using the Q system, and these computed values were then compared with the direct pressure measurements at the roof beams close to the location and with pressures inferred from the height of loosened rock (based on the extensometer data). A relatively close match was forced in the back-analysis of the Donkin data in order to "peg" the parameter values presented in Table 4.2.

For the ch. 1889 situation for example, the support pressure for a Q value of 2.7 with a J_r of 1.9 and a J_n of 12 can be calculated from Barton's (1974) empirical equation:

$$P = \frac{2 J_n^{1/2} (Q)^{-1/3}}{3 J_r} \quad (kg/cm^2) \qquad (4.6)$$

where P is the support pressure:

i.e.,

$$P = \frac{2(12)^{1/2}\,(2.7)^{-1/3}}{3(1.9)}$$

$$= 0.87\ \mathrm{kg/cm^2}$$ (see Table 4.1)

or 85.4 kPa

As an alternative to calculating this using equation (4.6), an approximation of support pressure can be directly derived from use of the upper graph in Figure 4.1. It should be noted however that variations in J_n ,the controlling block or slab size parameter, are not specifically taken into account in this graph as, for most situations, the relationship is relatively insensitive to this parameter.

Nevertheless, using either method, the value determined for ch 1889 correlates reasonably closely to the mean value between the measured pressure of 110 kPa (based on pressure cell readings) and that of 65 KPa inferred from calculations using the 2.5 m of loosened height estimated from the displacement measurements at this location (Yuen et al 1987). It should be noted, however, that as part of the methodology for modifying the parameter values, the values of J_a and J_r were iteratively refined within the range appropriate for the description of the rock at each location until a reasonable correlation was achieved between the empirically estimated support pressure at a given location and that considered most representative based on the pressure cell records or inferred from the displacement measurements at the tunnel roof.

This same approach was used for verifying the other descriptor codes for the various Q system parameters, until reasonable agreement was achieved by such back-analysis at all the instrumented locations along the tunnel. The values for the parameters finally developed for each of the

Q system components for each of the examined locations are listed in Table 4.1. The stratigraphy at each of the selected locations is also summarized on the same table together with the support pressures estimated from Barton's empirical relations. In each case, the estimated support pressures were matched against the pressures measured on the ring beams or inferred from the displacement measurements from the crown extensometers.

Throughout this back-analysis appraisal, parameter values for input to the classification were selected on the basis of the following conditions:

- the combination of the input parameter values used would provide a close match between the estimated and measured (or inferred) support pressures, and
- the values of the parameters would be appropriate for describing the rock mass characteristics at the sites examined.

As this process was iterative in order to obtain the best match between the estimated support pressure inferred empirically from the Q index values with the actual pressures deduced from the instrumentation, the descriptors used for the parameters were each individually modified during the back-analyses such that, after completion, they generally were appropriate to reflect the characteristics of the coal measures strata encountered at each location (Table 4.1).

In this regard, it is noteworthy that the overall match between estimated and measured pressures is generally acceptable in view of the empirical nature of the support pressure estimate and the need to maintain an appropriate balance between the ranges of selected parameters and the rock mass descriptors under the Q system.

As a result of the back-analyses carried out for the Donkin Mine, a set of appropriate input parameter values for the Q index components were established for describing typical coal measures strata in the Sydney Coalfield (see Table 4.2). This set of values can be considered as the basis for utilizing the Q classification rock mass conditions in the other operating coal mines.

4.4.2 Parameters for the CSIR-RMR System

Back-analyses of the same instrumented study areas in the Donkin Tunnels was also undertaken by using the CSIR-RMR classification system (Figure 4.7). As with the Q system, modifications to the original set of the component classification descriptors proposed by Bieniawski (1973) have had to be made in order to effect a reasonable match between pressures empirically assess from the classification with those measured or inferred from the pressure cell and extensometer data. Again, for illustration of the back-analysis procedure adopted, the extensometer location at Ch. 1889 is utilized.

RQD Parameter

Where available, drill core quality derived RQD values (refer to the discussion in section 4.4.1) can be used directly as input for computing RMR system ratings. Where no core data is available, however, as, for example, at Ch. 1889, Figures 4.4 and 4.5 can be used to relate mean discontinuity spacing or fracture frequency to RQD. Accordingly, the RQD at ch 1889 was assessed from Figure 4.5 as 60%. From the original RMR classification system ratings (Table 2.2) or from Figure 4.8 which provides a generic curve allowing determination of intermediate RQD values, a rating of 13.0 is selected.

Discontinuity Spacing Parameter

It is important to realize that the terminology used in the original RMR system addresses discontinuities in hard rock. In order to be consistent with worldwide standard bedding thickness descriptions, minor modifications have, therefore, had to be made to the descriptor codes and hence to the discontinuity spacing descriptor ratings.
As the critical discontinuities in the Sydney Coalfield are generally the bedding planes, the controlling parameter ratings and codings for discontinuity spacing definition have been modified to reflect bed thickness variations rather than jointing (see Table 4.4). For the Ch. 1889 example, as experience suggests that an average bed thickness value for beds within the potentially loosened zone should be used for assigning this parameter a mean discontinuity spacing of 100 mm has been selected from the field data. For this bed spacing, a corresponding modified RMR discontinuity spacing rating of 8.0 is ascribed (Table 4.4).

Condition of Discontinuity Parameter

This parameter describes the condition of the controlling discontinuities. It therefore allows assessment of both the surface condition and the alteration or infilling present on the discontinuities. As with the Q-system, the surface condition descriptors used for the original hard rock classification approach are directly transferable in application to horizontally bedded strata, however, the alteration and infilling descriptions require some modifications to reflect the nature of the major discontinuity surfaces (the bedding planes). To maintain consistency with the original intent of the CSIR-RMR ratings for this parameter, the overall range of possible values have been held constant within the bracketed values presented in Figure 2.2. However, the descriptions within each of the specific items have been adjusted to more closely reflect conditions pertaining to coal measures rocks.

The modified descriptors so derived are shown in Table 4.4. For the Ch. 1889 m example, based on field assessment and core description of the contact zone between the interbedded siltstone/sandstone unit and the overlying mudstone unit, a RMR condition rating of 20 was selected.

Intact Rock Strength Parameter

Intact rock strengths are usually determined from laboratory uniaxial compressive strength tests or from point load strength tests and they are independent of the hard or soft rock terminology. The rating values presented in Table 4.4 for this parameter do not require modification from the original RMR ratings (Figure 2.2) as they are still directly applicable. Thus, for the Ch. 1889 example, a rating of 6 (interpolated between the values given in Table 4.4, refer to Figure 4.8) was assigned corresponding to a uniaxial compressive strength value of 80 MPa.

Groundwater Parameter

As with the Q system, because the influence of groundwater is essentially identical in a hard rock or soft rock environment (provided that the rock material is not softened appreciably by the presence of water) the groundwater parameter as selected from the original CSIR-RMR rating tables (Figure 2.2) is independent of rock type and therefore does not require modification. For Ch.

1889 m, a groundwater rating of 15.0 was chosen reflecting the dry to slightly damp conditions identified at the site.

CSIR-RMR Rating

The resulting RMR for the Ch. 1889 situation was then calculated as follows:

$$\text{RMR} = 13 + 8 + 20 + 6 + 15 = 62$$

Support Pressure Correlation

A support pressure of 110 KPa was measured by the pressure cell installation at ch. 1889 m. The predicted support pressure based on an RMR of 62 and the expression (equation 4.1) suggested by Unal (1983) for calculating pressures from RMR values is as follows:

$$P = \frac{(100 - \text{RMR})}{100} \cdot \gamma \times B \tag{4.1}$$

$$= \frac{(100 - 62)}{(100)} \cdot (26 \text{ kN/m}^3) \cdot (7.6 \text{ m})$$

$$= 75 \text{ kPa}$$

This predicted pressure, like that decuded from the Q system is slightly less than that measured at the pressure cell (110 KPa) but slightly higher than the pressure inferred from the extensometer displacement data (65 KPa).

Similar matches were not always achieved when the other instrumented sections were initially evaluated. However, as with the Q system, the procedure described above was recursively utilized at each of the other study areas along the tunnel until good matches were achieved for all cases. Once this was done, specific refinements of the discontinuity condition terms were carried out for each classification parameter. The final ranges of parameter value descriptor codes established by this iterative procedure (Figure 4.7) and considered valid for characterizing coal measure sedimentary rock units with the CSIR-RMR classification system are presented in Table 4.4.

4.4.3 Parameters for the RSR Systems

The input parameters ranges for the RSR system are much less stringent in definition than those for the two other methods already discussed. In fact, they can generally be chosen by reference to the parameter values appropriate for defining the Q and/or RMR indexes. Hence, no specific back-analysis appraisal was needed for defining the parameter values for this system. Some minor modification of terminology of the original descriptor codes has been necessary though. These modifications are illustrated by application of the RSR system to classifying the coal measures strata for the rock conditions at Ch. 1889 m in the Donkin mine. The method of application and the appropriateness of the input parameters are discussed below:

Input for the RSR classification system consists of the following three parameters:

- Parameter A - general area geology
- Parameter B - joint pattern and direction of drivage
- Parameter C - groundwater and joint conditions

Parameter A, which assesses the rock mass in terms of rock type, hardness and geological structure requires careful use in practise, but does not require modification for application to Sydney Coalfield rocks. Assessment Values for parameter A are presented in Figure 2.3. For the conditions at ch 1889 m a value of 27 was chosen for parameter A assuming hard sedimentary rock (Type 2), of generally undeformed (massive) character.

Parameter B (Figure 2.3) provides a measure relating the discontinuity spacing and direction of tunnel drivage. As the major discontinuities in the Sydney Coalfield are bedding planes, application of the specific discontinuity spacing terms need some consideration in order that the assigned values reflect average bed thicknesses. As the original descriptors were expressed in terms of hard jointed blocky rock, these have been modified to reflect sedimentary bed thicknesses. The direction of drive terms, however, remain unchanged. Values for parameter B including the modified spacing terms in Figure 2.3 are presented in Table 4.5.

For the case of gently dipping (initially flat) coal measures strata with a discontinuity spacing of about 100 mm, the value of Parameter B is selected to be 13.

The final input parameter, parameter C, allows description of groundwater conditions with reference to ascribed discontinuity characteristics. For this part of the classification, the groundwater descriptors are directly applicable and required no further modification. The discontinuity condition descriptions although very general, are extremely subjective, making use of the terms good, fair and poor; these therefore also required no modification. Both components of the parameter C table shown in Figure 2.3 are applicable to the coalfield rocks. For the conditions at Ch. 1889 m, with no water inflow and fair discontinuity conditions, a value of 18 was selected for Parameter C.

In addition to the three parameters: A, B and C described above, an adjustment factor for the use of a tunnel boring machine for tunnel excavation was introduced into the RSR system to adjust the rating in accordance with the excavated diameter of the tunnel (see Figure 2.4). For the tunnels at Donkin, the adjustment factor is 1.14.

Thus, for ch 1889, the final adjusted RSR value is:
1.14 (A+B+C) = 1.14 (27+13+18) = 66.

As with the Q and RMR systems, the procedure described above was also used for the other selected instrumented areas along the Donkin tunnel in order to evaluate RSR ratings at each location. The parameters used and classification results are summarized in Table 4.1.

4.4.4 Summary

As none of the original classification systems were developed with data from stratified sedimentary rocks, the methodology described above was used to provide the initial reference data set from which to develop a set of parameter descriptor terms that would address the specific nature of coal measures strata including the rocks of the Sydney Coalfield.

The iterative parametric evaluation was therefore undertaken as a means of "pegging" the existing classifications to known rock behaviour. This has allowed both rational categorization and development of appropriate ranges of representative parameter values for the general range of Sydney Coalfield rock conditions.

Since the Donkin Tunnels intersect most of the rock units encountered in the operating coal mines, it was considered that this back-analysis should be undertaken on the Donkin data, assuming that the various rock conditions at Donkin are reasonably representative of rock conditions elsewhere in the coalfield. Further, because the support installation procedures and practices at Donkin were well documented, a reasonable correlation between rock mass classification and pressures could be made. Consequently the direct application of these parameter values in the other mines provides a means for evaluating and checking the applicability of the suggested methods.

As discussed in later sections, there are further adjustments that may well be necessary to each of the classifications in order to account for different support installation procedures and other factors not currently covered by the rock classification systems. As an illustration of such adjustment procedures, one particular modification/derating approach for assessing rock conditions in the gate roads adjacent to retreating longwall faces has been developed at this time. This is further discussed in Section 6.1.

Table 4.1 - Rock Mass Classification for the Donkin-Morien Tunnel

TUNNEL CHAINAGE		1428m	1889m	2263m	2433	2749	2996	3205	3385	3457	3522
INPUT PARAMETERS											
Span	(metres)	7.6	7.6	7.6	7.6	7.6	7.6	7.6	7.6	7.6	7.6
Height of loosened rock (from extensometer measurements)	(metres)	4	2.5	3	2.8	1.7	2.7	2.5	2.2	3.1	1.5
Roof Type (see Figure 5.1)		E	F	E	C	F	F	F	F	E	F
RQD	(%)	55	60	50	70	75	40	50	50	40	90
Discontinuity spacing	(mm)	100	100	100	200	200	100	100	90	80	300
Compressive strength	(MPa)	60	80	36	50	75	40	65	60	45	100
Groundwater flow per 10m	(l/min.)	0	0	0	0	0	0	0	0	0	0
Peak friction angle	(degrees)	25	28	23.5	23.5	25	24	25.5	25	25.5	35
Residual friction angle	(degrees)	17.5	22	16.5	15	22	16	19	20.5	22.5	30
Average block size	(cm)	3	5	3	3	15	3	5	5	5	25
CLASSIFICATION PARAMETERS											
NGI - Q											
RQD (from borehole or fracture spacing data)		55.0	60.0	50.0	70.0	75.0	40.0	50.0	50.0	40.0	90.0
Jn (calc. from block size)		18.3	12.0	16.7	23.3	5.0	13.3	10.0	10.0	8.0	3.6
Jr (calc. from Ja & peak friction angle)		2.9	1.9	3.1	3.9	1.7	3.4	2.4	2.0	1.6	1.2
Ja (calc. from residual friction angle)		6.2	3.6	7.2	9.0	3.6	7.7	5.1	4.3	3.4	1.7
Jw (calc. from water inflow data)		1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
SRF		1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
CSIR - RMR											
RQD rating		12.1	13.0	11.2	14.6	15.5	8.8	11.2	11.2	8.8	19.0
Discontinuity spacing rating		8.0	8.0	8.0	8.0	10.0	10.0	8.0	8.0	8.0	10.0
Joint condition		15.0	20.0	12.5	15.0	25.0	17.5	25.0	20.0	17.5	30.0
Compressive strength rating		4.8	6.2	3.4	4.2	5.8	3.6	5.1	4.8	3.9	8.0
Groundwater rating		15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0
Orientation rating		0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
RSR											
Geology/structure rating		24	27	24	24	27	24	24	24	24	27
Discontinuity spacing/orientation rating		13	13	13	23	23	13	13	13	13	30
Groundwater/condition rating		18	18	18	18	22	18	22	18	18	25
Correction for TBM tunnel		1.14	1.14	1.14	1.14	1.14	1.14	1.14	1.14	1.14	1.14
FINAL ROCK MASS RATINGS											
Q		1.4	2.7	1.3	1.3	7.0	1.3	2.4	2.3	2.4	17.5
RMR		55	62	50	57	71	55	64	59	53	82
RSR		63	66	63	74	82	63	67	63	63	93
SUPPORT PRESSURE CALCULATIONS											
Support Pressure (from loosened height)	kPa	104.0	65.0	78.0	72.8	44.2	70.2	65.0	57.2	80.6	39.0
Support Pressure (from pressure cells)	kPa	40.0	110.0	---	---	---	---	---	470.0	---	---
Support Pressure (from Q)	kPa	86.1	85.4	78.3	73.7	45.6	63.0	63.4	78.4	85.1	39.9
Support Pressure (from RMR)	kPa	89.1	74.9	98.7	85.3	56.8	89.0	70.6	81.1	92.3	35.6

TABLE 4.2

MODIFIED NGI - Q ROCK MASS CLASSIFICATION SYSTEM DESCRIPTORS

PARAMETER	DESCRIPTION		VALUE
J_n	Bedding	Plane Spacing	
	Very thickly bedded	>600 mm	0.5
	Thickly bedded	200-600 mm	1.0
	Thinly bedded	100-200 mm	5.0
	Very thinly bedded	60-100 mm	10.0
	Laminated	20- 60 mm	20.0
	Thinly laminated or completely fissured (with listric surfaces)	<20 mm	30.0
J_r	Noncontinuous discontinuities		4.0
	Rough or irregular, undulating		3.0
	Smooth, undulating		2.0
	Slickensided, undulating		1.5
	Rough or irregular, planar		1.5
	Uneven to flat, planar		1.2
	Smooth, planar		1.0
	Slickensided, uneven to planar		0.8
	Slickensided, polished		0.5
J_a	Contact between similar lithological units		
	- sandstone/sandstone (coarse grained)	$(\phi_r=37.5^\circ)$	1.0
	- sandstone/sandstone (fine grained)	$(\phi_r=35^\circ)$	1.2
	- siltstone/siltstone	$(\phi_r=30^\circ)$	1.5
	- mudstone/mudstone	$(\phi_r=22^\circ)$	3.5
	Contact between sandstone-sandy siltstone units	$(\phi_r=28^\circ)$	2.0
	Contact between sandstone-siltstone units	$(\phi_r=25^\circ)$	3.0
	Contact between siltstone-mudstone	$(\phi_r=23.5^\circ)$	3.0
	Contact between sandstone-mudstone units	$(\phi_r=21^\circ)$	4.0
	Contact between sediments and coal seams (rider seams)	$(\phi_r=19^\circ)$	5.0
	Contact between sediments and seat earth type carbonaceous units	$(\phi_r=18^\circ)$	6.0
	Contact between slickensided mudstones/ carbonaceous listric surfaces	$(\phi_r=16^\circ)$	8.0
J_w	Dry to minor flow (<5 l/min)		1.0
	Medium inflow (occasional outwash of infilling)		0.66
	Large inflow, unfilled joints		0.5
	Large inflow, outwash of infilling		0.3
	Very high transient inflow		0.2-0.1
	Very high continuous inflow		0.1-0.05
SRF	Medium stress (virgin state), tight discontinuities		1
	Loose rock with open discontinuities (med stress)		5
	Moderately high stress (intersection areas) with loose rock developed in crown areas		8
	(low modulus rock units)		10
	High stress (adjacent to advancing longwall) (high modulus brittle rocks)		30

TABLE 4.3

ESTIMATE OF EFFECTIVE SHEAR STRENGTH ϕ VALUES FROM THE PARAMETERS J_r AND J_a

(a)

Rock wall contact Descriptions	J_r	$\tan^{-1}(J_r/J_a)^\circ = (\phi')$				
		J_a=0.75	1.0	2	3	4
A. Discontinuous joints	4	79°	76°	63°	53°	45°
B. Rough, undulating	3	76°	72°	56°	45°	37°
C. Smooth, undulating	2	69°	63°	45°	34°	27°
D. Slickensided, undul.	1.5	63°	56°	37°	27°	21°
E. Rough, planar	1.5	63°	56°	37°	27°	21°
F. Smooth, planar	1.0	53°	45°	27°	18°	14°
G. Slickensided, planar	0.5	34°	27°	14°	9.5°	7.1°

(b)

Rock wall contact when sheared Descriptions	J_r	$\tan^{-1}(J_r/J_a)^\circ = (\phi')$				
		J_a= 4	6	8	12	
A. Discontinuous joints	4	45°	34°	27°	18°	
B. Rough, undulating	3	37°	27°	21°	14°	
C. Smooth, undulating	2	27°	18°	14°	9.5°	
D. Slickensided, undul.	1.5	21°	14°	11°	7.1°	
E. Rough, planar	1.5	21°	14°	11°	7.1°	
F. Smooth, planar	1.0	14°	9.5°	7.1°	4.7°	
G. Slickensided, planar	0.5	7°	4.7°	3.6°	2.4°	

(c)

No rock wall contact when sheared	J_r	$\tan^{-1}(J_r/J_a)^\circ = (\phi')$				
		J_a=	6	8	12	
A. Disintegrated rock and clay	1.0		9.5°	7.1°	4.7°	
		J_a=	5			
B. Bands of silty- or sandy clay	1.0		11°			
		J_a=	10	13	20	
C. Thick continuous bands of clay	1.0		5.7°	4.4°	2.9°	

Adapted from Barton et al (1974).

TABLE 4.4

MODIFIED CSIR-RMR ROCK MASS CLASSIFICATION SYSTEM

PARAMETER	DESCRIPTION		VALUE
Intact Rock Strength (UCS)	very high	>250 MPa	15.0
	high	100 - 250 MPa	12.0
	medium	50 - 100 MPa	7.0
	moderate	25 - 50 MPa	4.0
	low	5 - 25 MPa	2.0
	very low	1 - 5 MPa	1.0
		<1 MPa	0
Drill Core Quality (RQD)	excellent	90 - 100%	20.0
	good	75 - 90%	17.0
	fair	50 - 75%	13.0
	poor	25 - 50%	8.0
	very poor	<25%	3.0
Discontinuity (bedding plane) Spacing	very thickly bedded	>2m	20.0
	thickly bedded	0.6 - 2 m	15.0
	medium bedded	200 - 600 mm	10.0
	thinly bedded	60 - 200 mm	8.0
	very thinly bedded	<60 mm	5.0
Discontinuity surface condition	Very rough, noncontinuous surfaces within similar lithological units		30.0
	Slightly rough surfaces between similar lithological units		25.0
	Slightly rough surfaces between interbedded rock units		20.0
	Planar to smooth surfaces within similar rock units		15.0
	Planar to smooth surfaces between interbedded rock units		10.0
	Slickensided surfaces within similar lithological units		5.0
	Slickensided surfaces between interbedded lithological units		3.0
	Contacts between sediments and carbonaceous muddy units		2.0
	Gouge filled partings		0
Groundwater (inflow/10 m)	Completely dry		15.0
	Damp (<10 l/min.)		10.0
	Wet (10-25 l/min.)		7.0
	Dripping (25-125 l/min.)		4.0
	Flowing (>125 l/min.)		0

TABLE 4.5

RSR ROCK CLASSIFICATION SYSTEM

MODIFIED PARAMETER EVALUATION SCHEME

Rock structure rating - Paarameter A: general area geology (after Wickham et. al 1974)

	Basic rock type				Geological structure			
	Hard	Med.	Soft	Decomp.	Massive	Slightly faulted or folded	Moderately faulted or folded	Intensely faulted or folded
Igneous	1	2	3	4				
Metamorphic	1	2	3	4				
Sedimentary	2	3	4	4				
Type 1					30	22	15	9
Type 2					27	20	13	8
Type 3					24	18	12	7
Type 4					19	15	10	6

Rock structure rating - Parameter B: joint pattern direction of drive
(modified from Wickham et al. 1974)

Average discontinuity spacing	Strike to axis Direction of drive					Strike to axis Direction of drive		
	With dip Dip of prominent discontinuities*			Against dip		Both Dip of prominent discontinuities*		
	Flat	Dipping	Vertical	Dipping	Vertical	Flat	Dipping	Vertical
1. Laminated <50 mm	9	11	13	10	12	9	9	7
2. Very thinly bedded 50-150 mm	13	16	19	15	17	14	14	11
3. Thinly bedded 150-300 mm	23	24	28	19	22	23	23	19
4. Medium bedded to blocky 300-600 mm	30	32	36	25	28	30	28	24
5. Blocky to massive 600-1200 mm	36	38	40	33	35	36	34	28
6. Massive >1200 mm	40	43	45	37	40	40	38	34

Rock structure rating - Parameter C: ground water, discontinuity condition (modified from Wickham et al., 1974)

Anticipated water inflow (gpm/300 m)	Sum of parameters A + B 13-44			45 - 75		
	Discontinuity conditions**					
	Good	Fair	Poor	Good	Fair	Poor
None	22	18	12	25	22	18
Slight <200 gpm	19	15	9	23	19	14
Moderate 200 - 1000 gpm	15	11	7	21	16	12
Heavy >1000 gpm	10	8	6	18	14	10

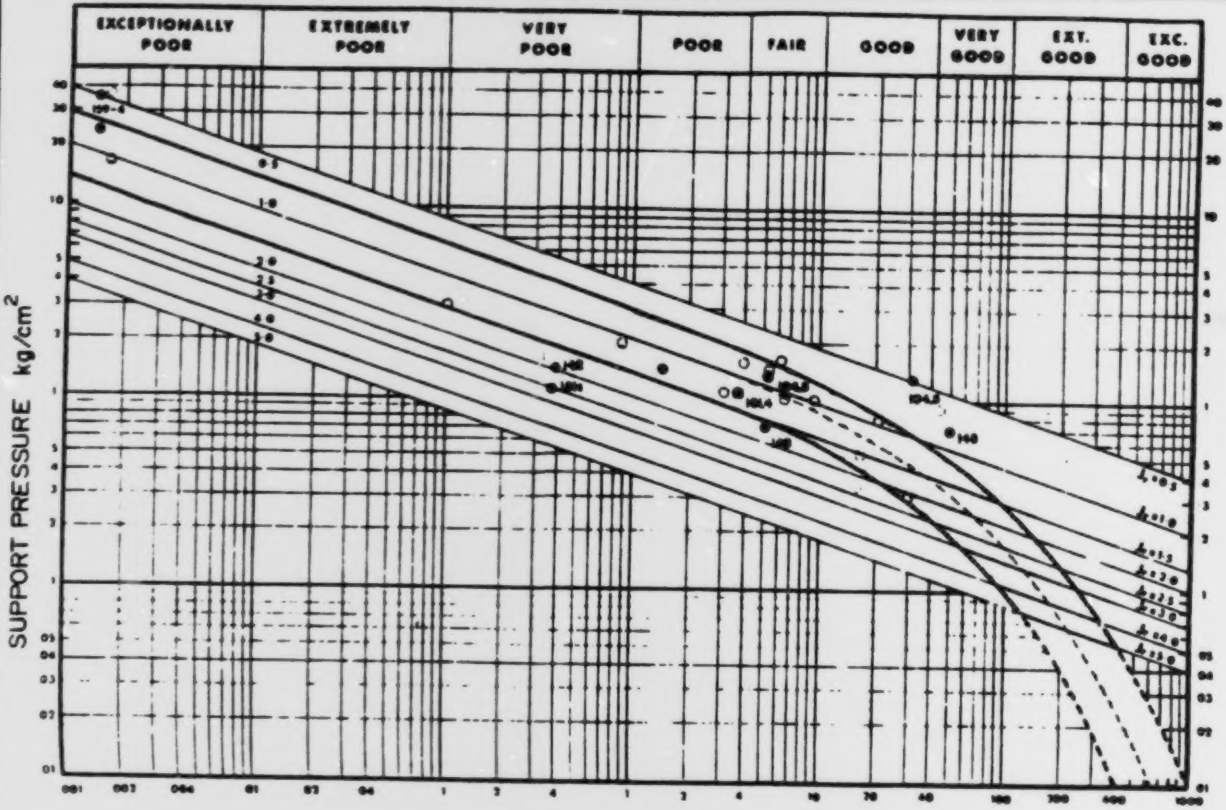
* Dip: flat: 0-20 deg; dipping: 20-50 deg; and vertical: 50-90 deg.

** Joint condition: Good = tight or cemented: Fair = slightly weathered or altered: Poor = severely weathered, altered or open.

ADAPTED FROM WICKHAM et al (1972) AND MODIFIED FOR PRIMARY USE IN SEDIMENTARY ROCK FORMATIONS

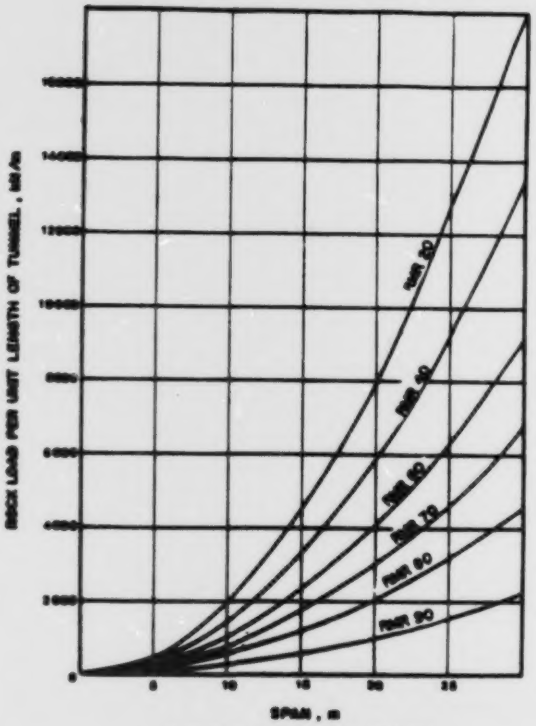
EMPIRICAL RELATIONSHIPS BETWEEN SUPPORT PRESSURE AND ROCK MASS QUALITY (Q AND CSIR CLASSIFICATION SYSTEMS)

FIGURE 4.1



ROCK MASS QUALITY $Q = \left(\frac{RQD}{J_n}\right) \cdot \left(\frac{J_r}{J_a}\right) \cdot \left(\frac{J_w}{SRF}\right)$

Adapted from Barton et al (1974)



Adapted from Unal (1983)

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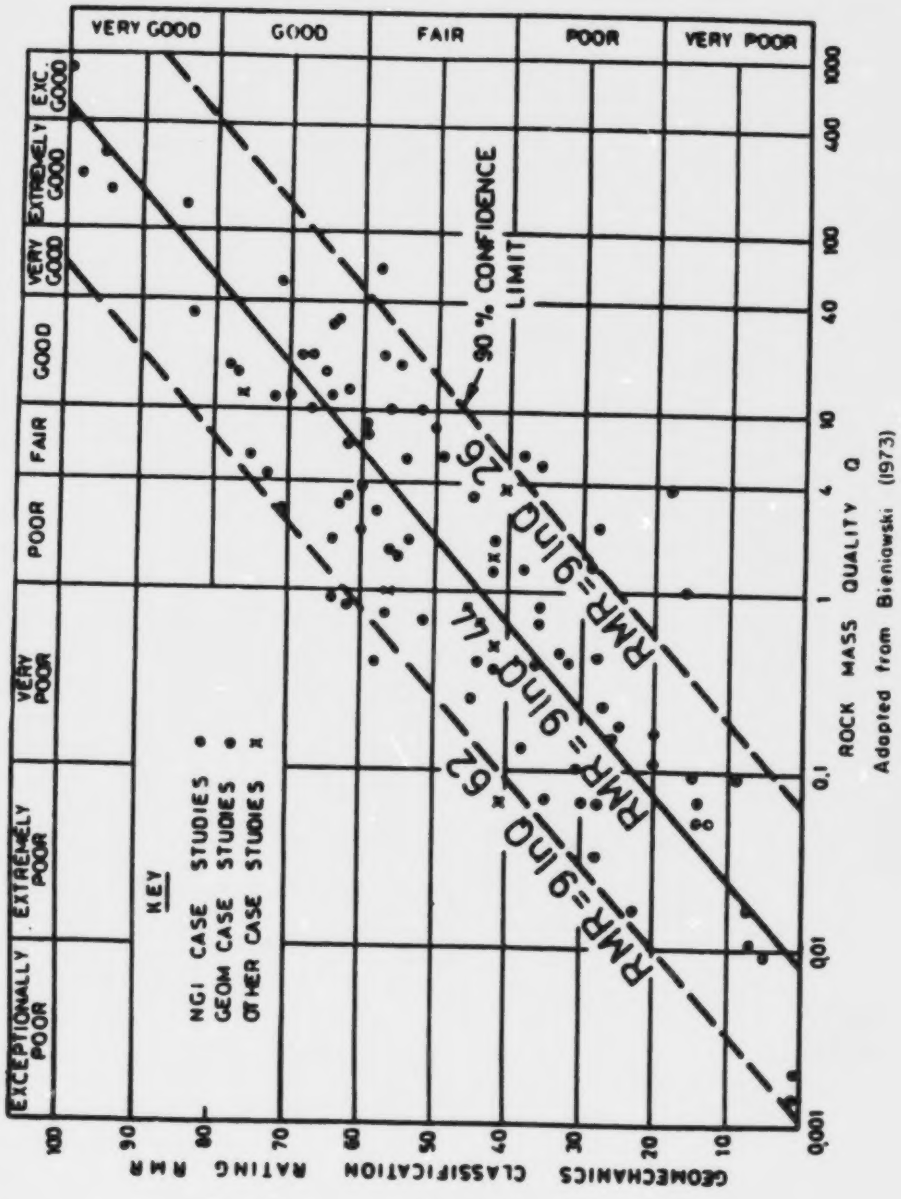
Drawn CPAM

Chkd

Form G.A.-D-4m (metric)

CORRELATION BETWEEN GEOMECHANICS RMR CLASSIFICATION AND Q SYSTEM

FIGURE 4.2



Adapted from Bieniawski (1973)

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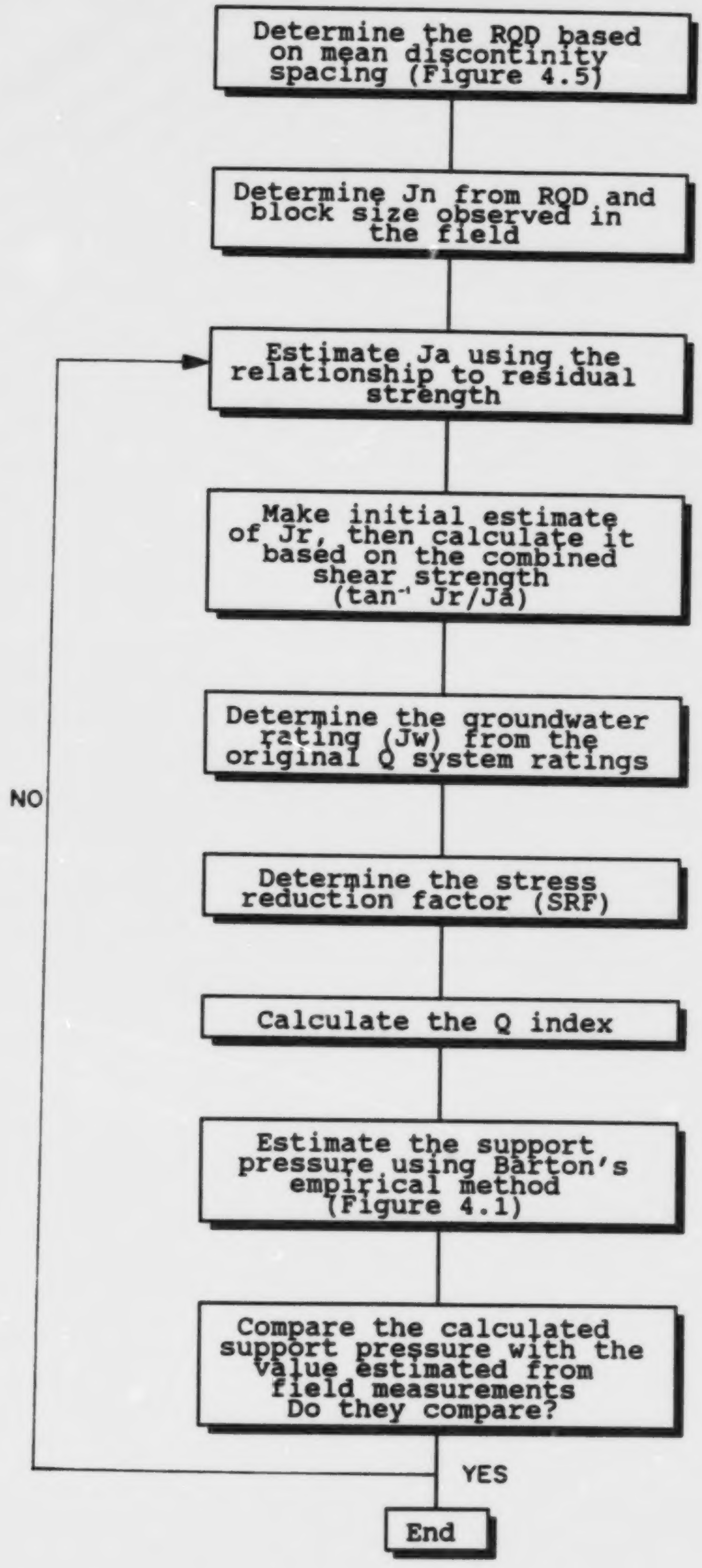
PROCEDURE FOR ESTABLISHING ROCK MASS DESCRIPTIONS BY BACK-ANALYSIS (Q SYSTEM)

FIGURE 4.3


Determine the RQD based on mean discontinuity spacing (Figure 4.5)
Determine Jn from RQD and block size observed in the field
Estimate Ja using the relationship to residual strength
Make initial estimate of Jr, then calculate it based on the combined shear strength ($\tan^{-1}$ Jr/Ja)
Determine the groundwater rating (Jw) from the original Q system ratings
Determine the stress reduction factor (SRF)
Calculate the Q index
Estimate the support pressure using Barton's empirical method (Figure 4.1)
Compare the calculated support pressure with the value estimated from field measurements Do they compare?
NO
YES
End

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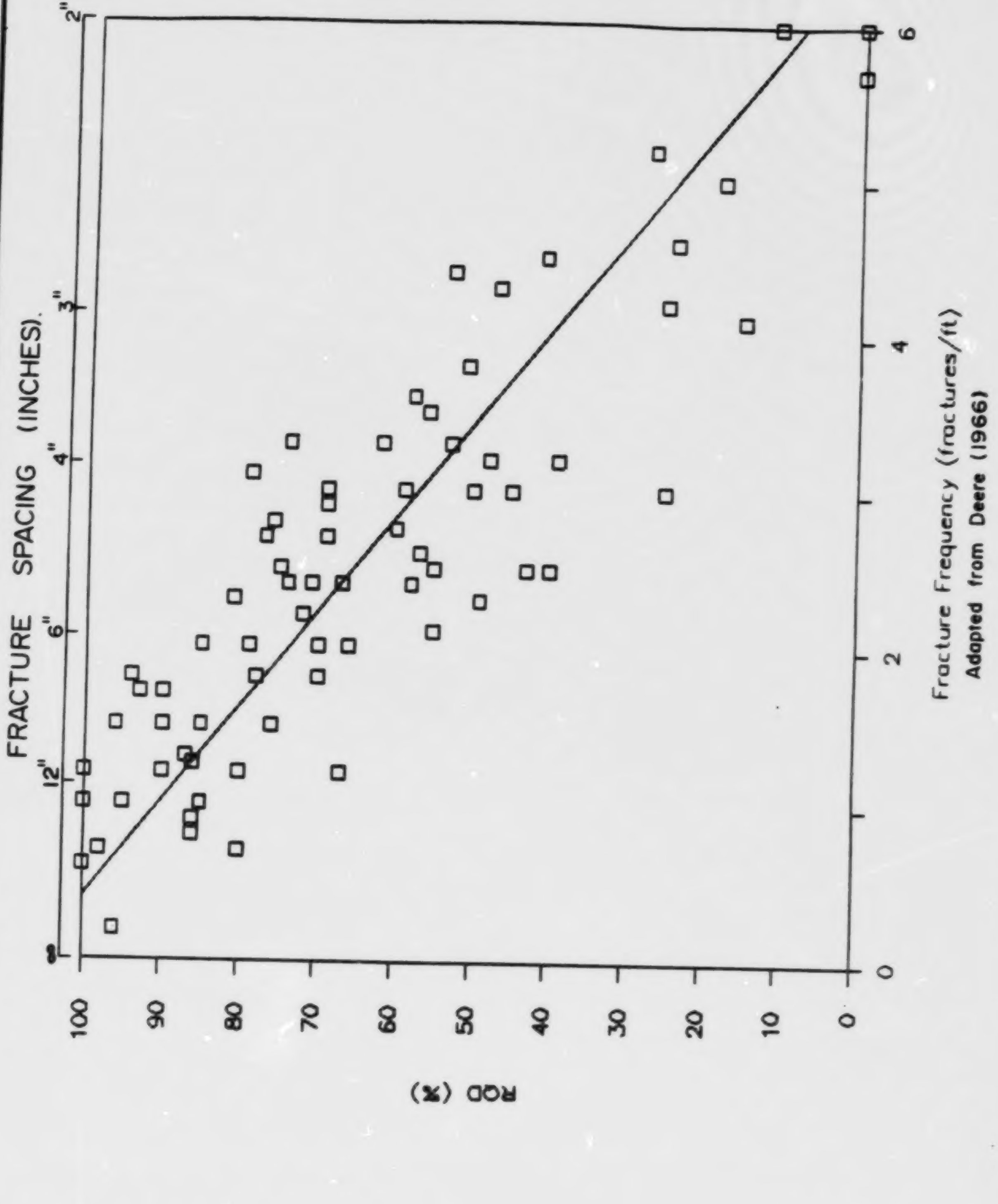
Chkd.

Form G.A.-U-4m (metric)

FRACTURE FREQUENCY VERSES RQD
(ADAPTED FROM DEERE, 1966)

FIGURE 4.4



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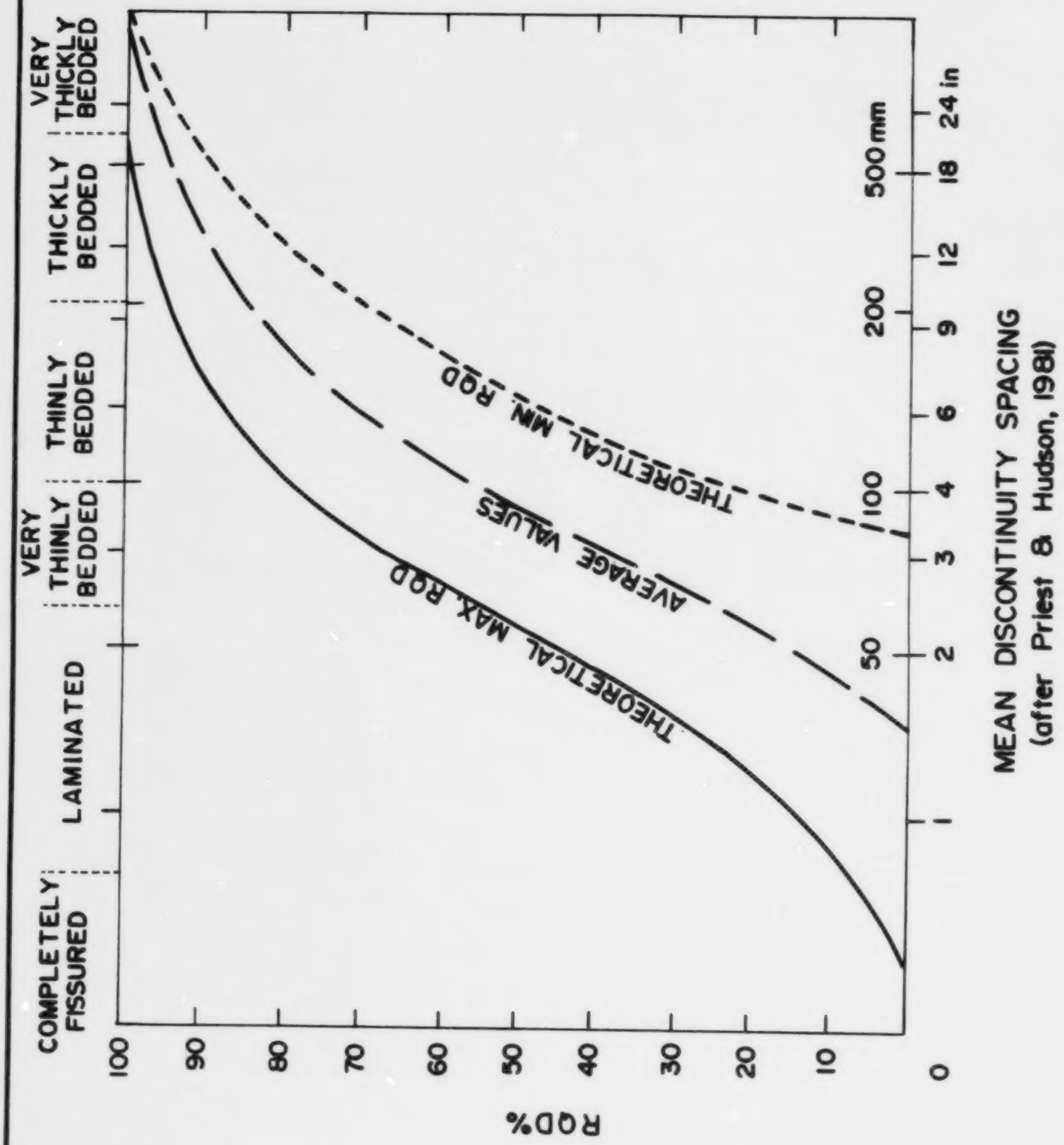
RQD VERSUS MEAN DISCONTINUITY SPACING

FIGURE 4.5



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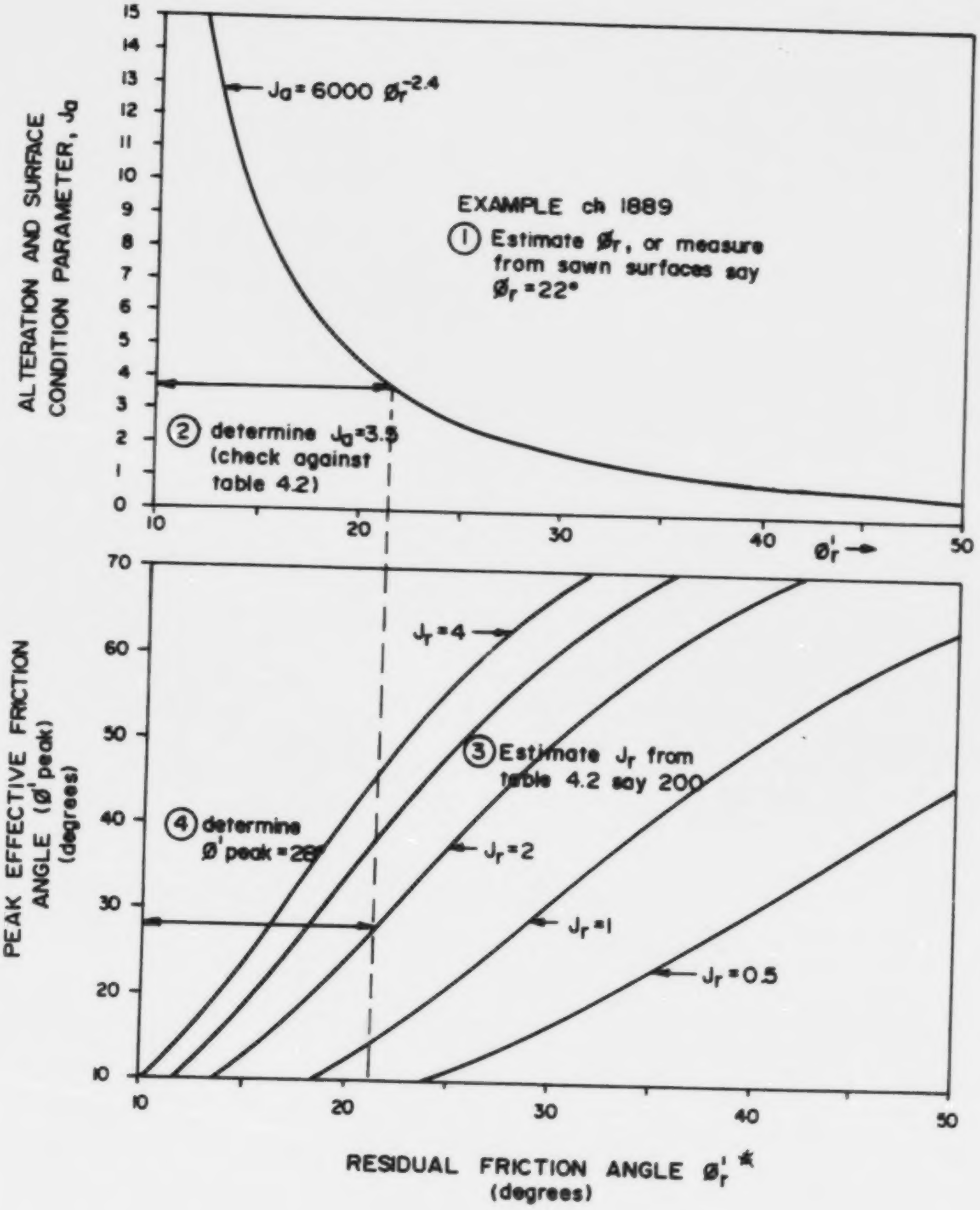
Chkd

APPROXIMATE RELATIONSHIPS BETWEEN J_r AND J_a AND PEAK AND RESIDUAL FRICTION ANGLES (MODIFIED FROM BARTON ET AL, 1974)

FIGURE 4.6



*obtainable by shear testing of sawn surfaces of rock or by inspection from table 4.2

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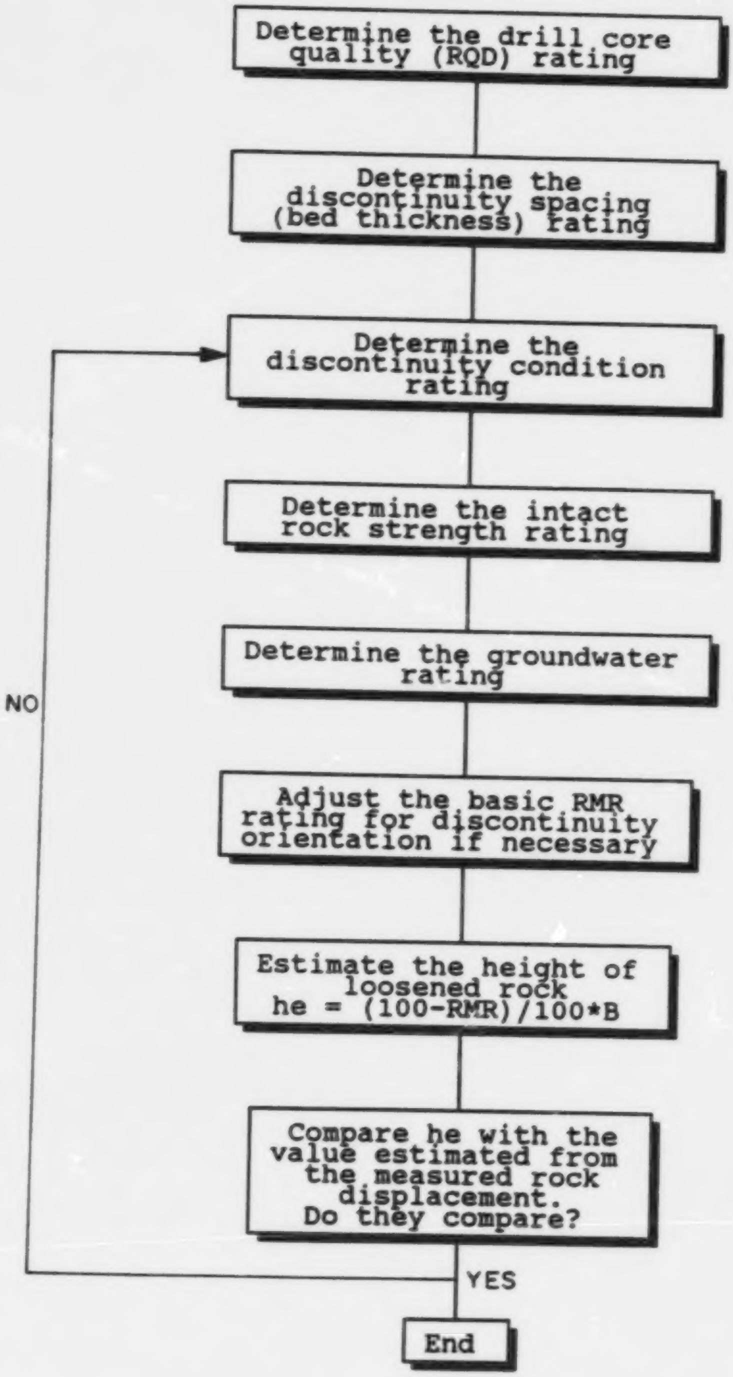
Chkd

PROCEDURE FOR ESTABLISHING ROCK MASS DESCRIPTORS BY BACK-ANALYSIS (CSIR SYSTEM)

FIGURE 4.7


Determine the drill core quality (RQD) rating
Determine the discontinuity spacing (bed thickness) rating
Determine the discontinuity condition rating
Determine the intact rock strength rating
Determine the groundwater rating
Adjust the basic RMR rating for discontinuity orientation if necessary
Estimate the height of loosened rock he = (100-RMR)/100*B
Compare he with the value estimated from the measured rock displacement. Do they compare?
NO
YES
End

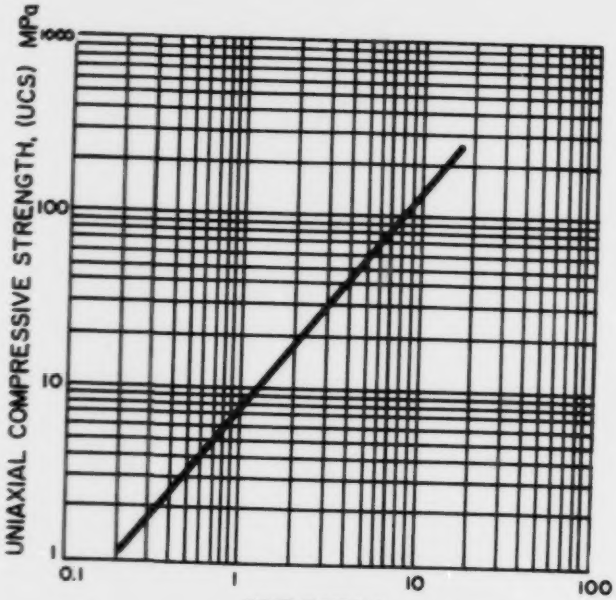
Date MARCH, 1989
Project 881-1719

Golder Associates

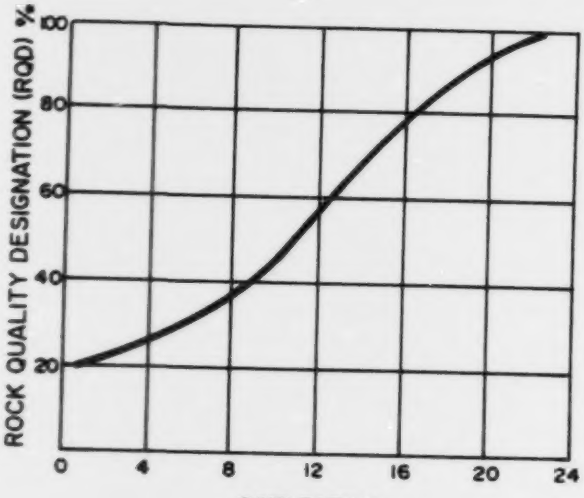
Drawn TDR
Chkd

CSIR ROCK MASS RATING VALUES FOR UNIAXIAL COMPRESSIVE STRENGTH AND RQD

FIGURE 4.8



U.C.S (σc)



RQD %

Date JUNE 1989
Project 881-1719

Golder Associates

Drawn KP
Chkd

5.0 APPLICATION OF ROCK MASS CLASSIFICATIONS TO THE SYDNEY COALFIELD

The revised classification descriptor tables and charts developed for coal measures rocks from the Donkin data are presented in Section 4.0 for the three rock mass classification systems studied. These tabulations provide a ready reference for the application of the Q, RMR and RSR classification schemes to the conditions experienced in the mines of the Sydney Coalfield.

In the following sections, selected locations in each of the accessible coal mines (the Donkin, Prince, Lingan and Phalen Mines) of the Sydney Coalfield have been classified to evaluate the validity of using these three classification schemes in the Sydney Coalfield. The locations that have been examined were chosen in consultation with CBDC geologists so as to cover a range of roof conditions within each of the Mines. Each of the selected areas in Prince, Lingan and Phalen Mines was visited and the following items were examined and documented:

a. exposed rock conditions including the rock type, bedding spacing and characteristics, the block size of loosened rock above the roof beam and on the side walls, jointing (other than bedding) and generalized groundwater conditions,

b. the type of installed rock support and its performance behaviour including a record of the geometry (horse-shoe, circular or rectangular) and size(s) of the structural members, and

c. approximate measurements of roof beam deflection (by stringing a line across the supporting ends of a roof support beam and measuring the deflection at mid span).

As the basis of the classification ratings had originated from data for the Donkin situation, this mine was not revisited.

For the other mines examined in this study, roof conditions were evaluated using the general scheme of roof geology categorization developed by Forgeron (1988) for the mines of the Sydney Coalfield. With some modifications, this roof geology categorization can be grouped into nine (9) types as shown in Figure 5.1. As these groupings of roof types are based on the system currently used by CBDC in the operating mines, a consistent reference with regard to geological mapping and the recorded conditions was maintained throughout the application of the classifications elsewhere in the coalfield. In order to apply the classifications, average mechanical properties of the rocks in the Donkin, Prince, Lingan and Phalen Mines were abstracted from published sources and from CBDC data. These are summarized in Table 5.1. The results of in situ stress measurements undertaken at each of these mines were also abstracted from published data (Canmet, 1988) and are summarized in Table 5.2.

For each of the mines visited, the conditions observed in the mines or inferred from available drilling and mapping data were tabulated. These summary tables were then utilized to form the basis for delineating roof type groupings for each particular selected location. Selection of parameter values for classifications was based primarily on this information and the data available for rock properties and in situ stress states. In order to provide a check on the veracity of the suggested classifications, the approximate rock loading on the support system at each location was assessed by back-analyzing the measured deflections of the roof beams.

The rock load acting on to the support system was back-analysed using a structural analysis program, PFRAME. The procedure involved back-calculating the applied rock load from the measured deflections, using various loading combinations. By this means, the equivalent depths of loosened rock and the pressures corresponding to the observed deflections were evaluated. These results were then compared with the approximate rock pressure empirically predicted by the Q and RMR classifications for the specific locations in question. The usefulness of the proposed classifications and the classification schemes could, by this process, be checked against both an intuitive assessment of rock mass quality and a practical evaluation of actual support loadings.

In order to allow comparison to the Donkin data, discussed at length in Section 4, the general geology and the classification results for the Donkin tunnels are summarized first followed by discussions of the application of the proposed system to the three other mines examined.

5.1 Rock Mass Classification for Donkin Mine

5.1.1 General

In Section 4 of this report, the Donkin TBM tunnel data was examined in some detail to develop a set of appropriate classification descriptors for later generic application of rock classification methods to other mines of the Sydney Coalfield. As part of this development procedure, recursive checking and cross-correlation was undertaken of each of the classification methods when applied to the rock mass conditions at each of the selected locations along the

Donkin TBM drivage. By this means the parameter values for each of the classification systems were iteratively refined until a reasonable match was achieved between the support pressure estimated from extensometer readings and that derived from the empirical relationships given by the classification schemes.

The range of parameter descriptors and the assigned numeric values as presented in Tables 4.3 and 4.4 for the Q and CSIR-RMR systems respectively constitutes the final set of refined tables of input parameters. These tables which can be considered reliably applicable for the Donkin mine then can be used to form the basis for classifying the coal measures rocks at the other operating mines throughout the coalfield.

5.1.2 Rock Conditions of Study Areas

Because of the cross-strata nature of the drivages at Donkin, most of the various coal measures rocks that are encountered throughout the coalfield were all intermediate by the tunnel drivage. Based on underground exposure and geological mapping of rock cores from the site, seven basic rock types were identified as follows:

ROCK TYPE	LITHOLOGICAL DESCRIPTION
I	Sandstone, fresh to slightly weathered, laminated to thickly bedded, light grey fine to coarse grained.
II	Interbedded sandstone/siltstone, fresh to slightly weathered, laminated to thickly bedded, light to dark grey argillaceous siltstone and fine grained sandstone frequently interlaminated and or gradationally bedded.
III	Siltstone, fresh to slightly weathered, laminated to thickly bedded light to dark grey argillaceous to fine grained sandy siltstone with occasional ironstone nodules.

IV	Interbedded siltstone/mudstone, fresh to slightly weathered, laminated to thickly bedded, light to dark grey mudstone and sandy siltstone, frequently interlaminated and/or gradationally bedded, mudstone frequently fissile.
V	Mudstone, fresh to slightly weathered, thinly laminated to medium bedded, light to dark grey mudstone frequently fissile.
VI	Carbonaceous mudstone, fresh to slightly weathered, thinly laminated to medium bedded, dark grey to black mudstone containing clasts or interbeds of black carbonaceous mudstone and/or thinly laminated coal.
VIA	Coal, fresh to faintly weathered, laminated to thinly bedded, bright black cleated coal, contains minor mudstone and clay partings.

This coding system is useful for summarizing the stratigraphy at each location along the TBM drivage where rock mass classification study was undertaken (Figure 5.2). These roof type codes shown in Figure 5.1 are listed in Table 4.1 for the ten instrumented locations used for developing the classification tables.

5.1.3 Excavation and Support Methodology

The tunnel drivage at Donkin where the classification back-analysis studies were undertaken was accomplished using a full face LOVAT 7.6 m diameter TBM. Support in the form of ring beams comprising 150 x 150 mm wide flange beams separated by 24 thrust blocks of 100 x 100 mm hollow structural steel sections were placed generally at 1.5 m centres. Mesh panels were placed between the sets to support the rock between the sets. The support was installed at about 8 m from the face and less than 1 m from the end of the TBM shield. By the use of the TBM, damage to the rock mass was virtually eliminated and thus the applied pressure measured at the instrumented sections of the ring beam support provides a reasonably accurate reflection of true rock load. This is evidenced by the fact that as the ring beam support began to take load, the rock displacement was gradually reduced until a relatively stable condition was attained.

5.1.4 Rock Mass Classification of Study Areas

In addition to back-analyzing the Q, RMR and RSR ratings for each of the study areas, forward calculations were also undertaken on an iterative basis to test the validity of the classification methodology. The rock mass at each of the ten areas studied along the TBM drivage was classified using the RMR, Q and RSR classification systems discussed in Section 4 where the procedure for developing the classification rating was presented in detail. Table 4.1 shows the rock mass rating of the study areas based on the three classification schemes.

For comparison purposes, four different support pressure evaluations are presented in Table 4.1. These include:

- a value for each location, back-calculated from the height of the loosened zone (inferred from the zone of active rock displacement),
- two values, each appropriately derived from the empirical methods proposed in the Q and CSIR-RMR systems, and
- a measured value from pressure cells installed on the ring beam supports at three of the instrumented locations.

In view of the back-calculation procedure used, the match between the support pressures estimated from the extensometer readings and those derived from the empirical relationships are generally close. However, the very high measured pressure at approximately chainage 3385 m when compared with that computed from displacement data and the classifications schemes is not consistent. The value measured is considered to be probably due to point loading as it was found to be so much larger than that derived from the displacements and the empirical predictions. In overall terms the general trend of the pressures deduced from Q index and CSIR-RMR values are reasonably consistent, as also are the quality indices with respect to that anticipated for the observed conditions.

The ranges of Q, RMR and RSR for the rock conditions observed along the TBM drivages at the Donkin Mine are summarized below:

Classification Indices	Range	Overall Rock Mass Quality *
Q	1.3-17.5	poor to good
RMR	50-82	fair to very good
RSR (adjusted for TBM)	63-93	-

* from Figures 2.2 and 2.3

It should be noted that the quality descriptions "poor" and "fair" for the Q and the RMR systems are not analogous. Reference to Figure 4.2 illustrates the divergence in terminology; the same rock in both systems being termed a different quality. Further, though there is no overall rock mass quality descriptor code given for final RSR ratings, the fact that several correlation charts are available between the RSR and RMR systems allows description using the same quality terminology as the RMR system.

5.2 Rock Mass Classification for Prince Mine

5.2.1 General

In order to classify the rock mass at Prince Mine, an examination of available records including roof geology plans, borehole records, wall prognosis sections and support designs was undertaken. Discussions were held with CBDC geologists and underground visits were made in order to gather specific data necessary for the rock mass classification.

General observations on roof conditions, geology and support performance were made throughout the mine during the underground visits. In addition, detailed measurements of support deformation, bed thickness and observations on discontinuity conditions, rock types, and groundwater conditions were undertaken at 20 different locations. Figure 5.3 shows the general mine layout, the locations of the study areas and the general classification of the roof geology based on CBDC records.

5.2.2 Rock Conditions of Study Areas

Prince Mine is located at Point Aconi near North Sydney. The mining development area extends approximately 3 km offshore in a northerly direction. The workable coal seam at the mine is the Hub seam which is approximately 2.2 m thick and dips gently to the north at 3 to 5 degrees. The coal seam is currently mined by retreat longwall mining techniques.

In general, the coal seam is overlain by a mudstone unit of variable thicknesses ranging in places from as little as 25 mm to about 4 m. Above the mudstone, a sequence of interbedded siltstone and sandstone and shale occurs.

Where Paleo-river sandstone channels are present above the mudstone layers they have created quite variable roof conditions. The various conditions have been grouped by CBDC geologists into six general roof types as illustrated in Figure 5.1 (roof types A through F).

In most part of the mine, the units are stratigraphically regular. However, the contact between the mudstone and the overlying interbedded siltstone and sandstone is generally weak even though the strength of the mudstone layer is comparable to that of the siltstone and sandstone units. The mechanical properties of the rock are presented in Table 5.1. Overcoring and hydrofracturing stress measurements have been undertaken in Prince Mine and the data are summarized in Table 5.2.

5.2.3 Excavation and Support Methodology

The main roadways and gate roads at Prince Mine are excavated within the coal seam with sometimes other strata being encountered in the roof. Excavation is undertaken by the use of Dosco Mark 2A and 2B roadheaders. The main roadways are supported by steel sets consisting of 80x100 mm x 21 kg/m arched sections spaced at 1 m centres. Timber lagging is used to block the steel set at the roof arch and on the sides. Steel corrugated sheets of 1.2 m x 2 m are installed behind the sets to cover the arch and the leg portions. The gate roads are supported by 150 x 150 mm x 37 kg/m square sets of 4.3 m span at 1 m centres. Laggings for the gate roads are similar to those for the main roadways.

Most of the twenty areas studied in detail are located along the gate roads on the east side of the existing mining area and some were along the exploration drift developed in the future mining area.

5.2.4 Rock Mass Classification of Study Areas

Each of the selected study areas at Prince was classified using the modified RMR, Q and RSR classification systems. The observed roof conditions at each of the study areas are summarized in Table 5.3 and the results of the rock mass classifications including input parameters and ratings for each of the sub-classification parameters are presented in Table 5.4.

The range of assessed Q, RMR and RSR values for the rock conditions at the study areas is summarized below:

Classification Index	Range	Overall Rock Mass Quality
Q	1.97-294.9	very poor to extremely good
RMR	35-81	poor to very good
RSR	47-83	--

In general, the rock mass quality described by the classifications is consistent with observed rock mass characteristics and roof conditions. Further, there appears to be good correlation between CBDC empirical experience and the assessed rock mass quality ratings. For example, roof type A, where massive sandstone occurs, has the highest rating. On the hand, for roof type F, where silty laminated mudstone occurs over the coal seam, the rating is the lowest, consistent with actual experience that this situation often leads to roof fall problems.

In general, there is consistency amongst all three classification systems, although because of differences in the parameters addressed by each of the classifications with each roof type, there are variations due to local conditions. This is reflected in the computed support pressures as evaluated from the empirical methods for the Q and RMR systems when compared with that back-calculated from the observed deflections.

With regard to the two empirical methods (Q and RMR), the support pressure for both the Q and RMR generally increase with a decrease in rock mass rating. However, the support pressures calculated from Q are generally lower than those from the RMR system when the rock mass quality is good to very good. When the rock mass quality becomes fair to poor, the support pressure predicted from the Q system is generally higher than that computed using the RMR approach. In fact, for very poor rock mass quality, the support pressure predicted by the Q system is unrealistically high reflecting the potential support needed for an unsupported roof where a large roof collapse or chimney cave-in situation is anticipated. In bedded sedimentary rocks where strata are more or less horizontal, there is a limiting cave height that is appropriate for a given geometry of excavation. This limiting cave-height provides a control on the maximum achievable loosened height. Unal's (1983) expression for loosened height above coal mine openings based on the RMR system incorporates this limitation and indicates that the maximum height of loosening is the span of the opening. Evans and Lawrence (1988) give examples which relate the height of loosening to the practicable maximum bulking of the fallen rock mass. For rectangular and wedge shaped voids, the following equations pertain:

Rectangular Collapse	Wedge Collapse	Bulking Factor
$H=h\left(\frac{B}{B-1}\right)$	$H=2h\left(\frac{B}{B-1}\right)$	$B = \frac{\text{density roof}}{\text{density collapse}}$

where H = collapse height above the seam pavement
h = extracted seam height, and
B = bulking factor as defined above

Based on these expressions and a nominal bulking factor, the maximum loosened height above the crown of the opening can again be seen to be of the order of the height of the extracted seam. However, these expressions do not take into account the quality of the rock as in Unal's approach based on the RMR system. However, they can be used to define an upper bound value for loosened height as a modifier of the loading suggested from application of the Q system.

Not only do variations exist in assessed support pressure because of differences between the various components of each of the classification systems. Significant differences also exist between support pressures calculated from the observed deflections of the steel frames within each roof type. For example, in roof type B (along the experimental drift), the observed roof beam deflections vary from zero (non-detectable) to as much as 150 mm. Hence the back-calculated pressures vary from 0 to about 92 kPa. At location P13, the observed deflection is as much as 380 mm. The corresponding back-calculated support pressure is therefore very high. Similar discrepancies occurred with other roof types. However, in general, the response predicted by the classifications is not too far off what intuitively is expected for the observed conditions. This is discussed in more detail in Section 6.

5.3 Rock Mass Classification for Lingan Mine

5.3.1 General

Classification of the rock mass at Lingan Mine was carried out in a manner similar to that described for Prince Mine. Following a detailed review of available records on the geology and support design, underground visits were made to examine roof conditions and support performance.

Measurements of support deformations, bed thicknesses, fabric spacing and observations similar to those described for Prince Mine were conducted at the selected study areas. Eleven study areas were selected and their locations are depicted on the general mine layout plan shown in Figure 5.4.

5.3.2 Rock Conditions of Study Areas

The Lingan Mine is located near New Waterford. The mining development area extends approximately 3.5 km offshore in a near northerly direction. The workable coal seam at the mine is the Harbour seam which is approximately 1.8 m thick and dips at about 13 percent to the north.

In general, the coal seam is overlain by a thin mudstone seam which is in turn overlain by a sequence of siltstone, sandstone and limestone. Below the coal seam, the strata consists mainly of interbedded siltstone and mudstone with occasional thin layers of sandstone. The average mechanical properties of the rocks at Lingan are summarized in Table 5.1 along with those for the other mines. Overcoring and hydro-fracturing stress measurements have been undertaken at the mine and the data are summarized in Table 5.2.

At Lingan, the main roadway (slope) dimensions are 6 m wide x 3.7 m high and they are developed to a horseshoe shape with the base of the coal seam at invert level. The roof of the roadway is in either mudstone or siltstone depending on the local variations in the thickness of the mudstone and siltstone layers. Based on available geological data, these mudstones could vary from 0.2 to as much as 1.5 m in thickness.

5.3.3 Excavation and Support Methodology

The main roadways at Lingan Mine were excavated in the past by the drill-and-blast method with the coal taken out first followed by excavation of the strata above. In recent years, the excavation method has been improved by utilization of road header machines. The roadways are generally supported by steel sets consisting of 114 x 127 mm x 29 kg/m arched sections spaced at 1 m centres. These sets are generally erected at about 3 m behind the face during development of the roadway.

Timber lagging has been used throughout the mine to block the space between the steel sets and the rock. Such blocking has generally been placed at the roof arch (approximately 20 to 50 degrees from the centre of the arch). Occasionally, the lagging was brought down the sides to the invert. This was done at intersections and in areas where substantial overbreak and loosening had occurred. In all cases, steel corrugated sheets of 1.2 m x 2 m are installed between the steel sets over the arched portion, mostly down to the spring-line but in some cases right to the invert.

5.3.4 Deformation Behaviour

Distress of some of the support at Lingan is more pronounced than at the other mines, essentially because of the effects of advancing longwall mining. In general, roof falls where they could be observed were generally of up to 1 m in height and of classic cathedral shape controlled by delamination and slabbing on bedding planes. Jointing in some cases is visible; and where present, such joints are frequently slickensided particularly in argillaceous units. Although such joints are ubiquitous in some areas, no specific evidence could be found to suggest that the joints kinematically control the geometry of the rock falls.

Where "massive" sandstone forms the crown of the tunnels the failure mode appears to be by spalling along pre-existing fractures, possibly accentuated by blasting, rather than as a result of discontinuity control. In this regard, it should be noted that the method of excavation with the roadheader, where sandstone is in the crown is to deliberately undercut the crown to induce slabbing failure of the sandstone. This avoids having to blast the sandstone down.

At the deepest point of the workings (no. 7 west bottom) immediately beyond the main slope barrier pillar, the road is subject to "heavy roof". Excessive convergence was experienced in this area, the steel arches showed severe buckling. Yielding "U" section arches were subsequently used this area and these supports showed less deformation presumably as a result of the yielding capability of the arches.

5.3.5 Rock Mass Classification of Study Areas

The selected study areas are located in the 9E and 7W bottom gate roads as shown in Figure 5.4. Four roof types (C, F, G, H in accordance with the general roof type designations) were encountered and descriptions of the roof conditions at each of the study areas are summarized in Table 5.5.

Results of the rock mass classifications for each of the study areas are presented in Table 5.6.

The range of Q, RMR and RSR for the rock conditions for all of the study sites is summarized below:

Classification Index	Range	Overall Rock Mass Quality
Q	2.8 to 81.6	poor to very good
RMR	44 to 70	fair to good
RSR	49 to 69	-

For all the areas examined throughout this mine, as similarly noted from the classification results for Prince Mine, the empirical support pressures calculated from the Q and RMR ratings show a trend of increasing support pressure with decreasing classification rating. Again, the pressures based on the RMR rating are generally higher for fair to good rock, while for fair to poor rock, the pressure calculated from the Q index is higher. The minimum support pressure is predicted for the locations where roof type G was encountered, where siltstone and sandstone channels exist locally, these areas appear to have the highest rock quality ratings.

As with the Prince mine data, the pressures back-calculated from the measured deflections of the installed support also varied markedly. In fact, in many cases they do not correlate with the predicted pressures from the rock mass classification systems. This is not surprising due to the type of support practices applied in this mine and to the fact that some of the support for the area studied comprises steel arches which had already been deformed due to loading effects from the adjacent longwall waste. Attempted correlations for this mine then must therefore be viewed with some caution as it is difficult to have a good reference to define the deflection of the beam at the crown, especially as in some cases, yielding arches were used. Thus, the observed deflections as indicated in Table 5.6 may not represent an accurate assessment of the true deflection of the steel sets nor of the actual effects of rock pressure acting onto the supports as a result solely of the excavation of the gate road. The deflections and back-calculated pressures, however, provide a guide to the required values of the SRF parameter for the Q system for assessing the effect of longwall removal and for assigning a derating scale for the RMR system for the same application.

5.4 Rock Mass Classification for Phalen Mine

5.4.1 General

The data collection procedure used for classifying the rock mass at Phalen Mine was the same as that used for the other mines (refer to Sections 5.2 and 5.3).

A total of 17 areas in the vicinity of the number 1 and number 2 west panels were examined in detail in addition to general observations made throughout the mine. The locations of the study areas are shown in Figure 5.5.

5.4.2 Rock Conditions of the Study Areas

Phalen Mine is located at New Waterford adjacent to the Lingan Mine. The mining development area extends approximately 1.3 km offshore in a northerly direction. The workable coal seam at the mine is the Phalen seam which is about 1.8 m thick on the west side and about 2.5 m to 3 m thick on the east side of the mine. The coal seam is currently mined by retreat longwall mining techniques.

In general, the coal seam is overlain by a successive sequence of mudstone, siltstone and sandstone. Delamination tends to occur between the sandstone and the mudstone. The presence of paleo-river sandstone channels often caused weak roof conditions. The mechanical properties of the rocks at Phalen are similar to those of the other mines (see Table 5.1). The in situ stress state is also considered to be similar, however, no in situ stress data is available for this mine.

5.4.3 Excavation and Support Methodology

The excavation sequence used at Phalen is similar to that used the other mines discussed. The main roadways at Phalen Mine are supported by steel sets of 80 x 100 mm x 21 kg/m arched sections spaced at 1 m centres. Timber lagging and steel corrugated sheets similar to those for the Lingan and Prince Mines are used with the steel sets. The gate roads are generally supported by steel sets of 114 x 127 mm x 29 kg/m sections, 3.5 m to 3.7 m wide by 2.7 m high.

According to the information provided by the operations personnel at the mine, the roof of roadways and their intersections are generally located in siltstone. Initially, in most cases, roof conditions are generally stable; little deflection occurs in the steel sets except at intersections where only minor deflections develop.

An extensometer has been installed into the rock above the crown of one of the tunnels at the intersection of 2 Angle Deep and 3 West Bottom. The extensometer, which was installed by the Cape Breton Coal Research Laboratory of CANMET, was located between the third and fourth roof beams (spaced at 1 m centres), some 2.6 m from the east side leg of the supporting steel frame. At the time of installation, the tunnel face was about 2.5 m ahead of the extensometer location, and arch support was being placed at about 3.0 m from the face. The extensometer was monitored soon after installation on a regular basis but was discontinued when the cable guide for the extensometer (flexible probe extensometer) was displaced by the rock movement and no further reading was possible. However, the available monitoring results obtained indicate that the majority (90%) of the movements occurred soon after installation and were confined to within the first 2 m of the roof line.

5.4.4 Rock Mass Classification of Study Areas

The general mine layout and the seventeen study areas at Phalen Mine are shown in Figure 5.5. The study areas are located along the west bleeder, no. 1 west top and bottom gate road and no. 2 west top gate road. At the time of the site visit, no. 1 west panel was being developed. Roof types A to H were encountered along these gate roads. Descriptions of the roof conditions at each of the study areas are summarized in Table 5.7 in the same format as used for other mines.

Results of the rock mass classifications for each of the seventeen study areas are presented in Table 5.8. The ranges of Q, RMR and RSR for the rock conditions at the study areas within the Phalen Mine are summarized below:

Classification Index	Range	Overall Rock Mass Quality
Q	0.9 to 491.6	very poor to exceptionally good
RMR	41 to 91	fair to very good
RSR	49 to 76	-

For all the areas studied, all of the classification systems reflect observed conditions. Where roof type A, the massive sandstone roof, occurs the highest ratings are calculated, whereas when roof type E occurs, where weak laminated mudstone is encountered above the crown, the lowest rating was indicated and on occasions, ground problems were experienced at the mine.

The rock mass classifications again appear to reflect fairly accurately the perceived behaviour of the rock at the roof of the mine openings and the estimated support pressures again generally increase with decrease in the classification ratings. As a range of roof types were encountered within the seventeen study areas, the estimated support pressures vary considerably. The support pressures back-calculated from the roof beam deflection also vary markedly between roof types. Based on the observed deflections, the back-calculated pressures do not closely match those assessed from the rock classification empirical relationship (pressures calculated from the support loadings actually range from zero to just over 85 kPa). This lack of correlation may reflect timing of installed support and the lack of control on delamination due to the passive nature of the support system.

TABLE 5.1

AVERAGE MECHANICAL PROPERTIES OF ROCK UNITS IN THE DONKIN, LINGAN, PRINCE AND PHALEN MINES

	Unconfined Compressive Strength (MPa)	Young Modulus (E) (MPa)	Poisson's Ratio ν	Tensile Strength (MPa)
DONKIN MINE				
Sandstone	92	24000	0.25	7.5
Interbedded Sandstone-Siltstone	121	17600	0.25	9.0
Siltstone	53	11000	0.25	3.5
Interbedded Siltstone-Mudstone	36	9000	0.25	3.7
Mudstone	36	5400	0.25	3.5
Carbonaceous Mudstone	16.6	3450	0.1	-
Coal seam	16.6	3450	0.1	-
PRINCE MINE				
Mudstone	20	20340	-	-
Siltstone	48.3	15800	0.16	-
Sandstone	41.3-48.3	17650	0.20	4.0
LINGAN MINE				
Coal	11.0	3450	0.1	-
Mudstone	20.6	15450	0.11	-
Siltstone	48.3	36550	0.20	-
Sandstone	79.0	30340	0.18	-
PHALEN MINE				
Mudstone	28	12700	-	3.4
Siltstone	50.6	16300	-	6.8
Sandstone	58.1	16200	-	5.5

TABLE 5.2

STRESSES MEASURED AT DIFFERENT MINES IN THE SYDNEY COALFIELD

Mine Site	Depth of[1] Test Site (m)	Method of Stress Measurement	Principal Stresses[2] (MPa) (Azimuth/Dip) (Degrees) σ_1	σ_2	σ_3	Stress Components[3] (MPa) $\sigma_{E\text{-}W}$	$\sigma_{N\text{-}S}$	σ_v	Overburden Stress (MPa)
Donkin Mine	165 (12)	overcoring	-	-	-	9.04	7.93	4.90	4.2
No. 26 Colliery	762 (90)	overcoring	26.3 (278/64)	24.05 (9/13)	23.2 (270/-25)	23.31	24.98	25.27	21.7
Prince	222 (20)	overcoring	11.8 (263/14)	9.54 (-7.8/5)	5.1 (64/-75)	10.76	9.51	6.00	5.5
		hydrofracturing	17.8-25.4 (274/0)	12.8-14.1 (6/0)	5.4-5.9 (-)	-	-	-	-
Lingan	593 (70)	overcoring	33.84 (88/4.0)	14.49 (4/-55)	12.21 (355/35)	33.73	13.76	13.05	14.3
		hydrofracturing	-	-	7.8/15.4 (-)	-	-	-	-

Notes:

(1) Depth below sea level, Figure in ( ) indicates depth of water.
(2) σ_1, σ_2, σ_3: Major, intermediate and minor principal stresses.
(3) $\sigma_{E\text{-}W}$, $\sigma_{N\text{-}S}$, σ_v: Stress components in E-W, N-S and vertical directions.

TABLE 5.3

OBSERVED CONDITIONS OF STUDY AREAS

AT PRINCE MINE

Location (Roof Type)	Roof Conditions	Groundwater Conditions	Remarks
P1 (B)	1-1.5 feet of grey slightly silty mudstone underlying a light grey, medium to coarse grained massive sandstone	generally damp	
P2 (E)	approx. 4 feet of grey, laminated, silty mud-stone underlying a 1 foot thick coal seam	generally damp with some minor dripping	
P3 (E)	approx. 1 foot of grey, laminated silty mudstone underlying a coal seam	generally damp with some minor dripping	
P4 (E)	approx. 1 foot of grey, laminated silty mudstone underlying a coal seam	generally damp with some minor dripping	
P5 (D)	>10 feet of grey, laminated mudstone with listric surfaces and some coal chards	generally dry	roof conditions based on observations and data from borehole PM-39
P6 (C)	approx. 10-15 feet of grey, locally silty mudstone with listric surfaces and some coal chards	generally dry	roof conditions based on observations and data from borehole PM-39
P7 (F)	approx. >10 feet of grey, silty laminated mudstone with a thin coal seam at ~1 foot from the roof	generally dry	roof conditions based on observations and data from borehole PM-39
P8 (B)	massive sandstone overlying a thin >2 foot thick laminated mudstone layer	generally damp with minor dripping	

Location (Roof Type)	Roof Conditions	Groundwater Conditions	Remarks
P9 (B)	0.1-0.5 feet of grey laminated mudstone underlying a massive sandstone unit	generally damp with some minor dripping	very little of roof visible
P10 (B)	0.1-0.5 feet of grey laminated mudstone underlying a massive sandstone unit	generally damp with some minor dripping	very little of roof visible
P11 (B)	approx. 3 feet of grey slightly silty, moderately competent mudstone gradational upper contact with light grey, med grained, massive sandstone with poorly developed bedding	generally dry	based on observations and data from PM-40
P12 (C)	approx. >10 feet of grey, locally silty mudstone with some listric surfaces and coal stringers	generally dry	based on observations and data from borehole PM-39
P13 (C)	approx. >10 feet of grey, locally silty mudstone with some listric surfaces and coal stringers	generally dry	based on observations and data from borehole PM-39
P14	approx. >3 feet of grey slightly silty, moderately competent mudstone; gradational upper contact with very thin siltstone unit underlying light grey med grained massive sandstone	generally dry to damp	based on observation and data from borehole PM-40
P15 (E)	approx. 4 feet of grey laminated, silty mudstone underlying a 1 foot thick coal seam	generally damp with some minor dripping	

Location (Roof Type)	Roof Conditions	Groundwater Conditions	Remarks
P16 (A)	approx. >10 feet light grey, fine to med grained massive near roof to laminated sandstone with minor siltstone	generally damp with minor dripping	based on observation and data from borehole PM-35
P17 (B)	thin mudstone layer underlying a massive sandstone unit	generally damp with minor dripping	difficult to see much of roof
P18 (B)	thin mudstone layer underlying a massive sandstone unit	generally damp with minor dripping	difficult to see much of roof
P19 (B)	thin mudstone layer underlying a massive sandstone unit	generally damp with minor dripping	difficult to see much of roof
P20 (D)	light grey, thinly laminated mudstone	generally dry	difficult to see much of roof
P21 (A)	massive sandstone unit	generally damp	difficult to see much of roof

Table 5.4 - Rock Mass Classification for Prince Mine

LOCATION		P21	P16	P8	P10	P1	P9	P11	P17	P18	P19
INPUT PARAMETERS											
Span	(metres)	4.3	4.3	4.3	4.3	4.9	4.3	4.3	4.3	4.3	4.3
Support Spacing	(metres)	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9
Beam deflection	(cm)	0	3	0	8	15	0	13	0	3	8
Load (calc. from deflection)	(kN/m)	0	145	0	400	740	0	640	0	145	395
Roof Type (see Figure 5.1)		A	A	B	B	B	B	B	B	B	B
Discontinuity Spacing	(mm)	600	300	300	300	300	300	300	300	300	300
Uniaxial Compressive Strength	(MPa)	47	47	47	47	47	47	20	47	47	47
Groundwater Flow per 10m	(l/min)	5	5	5	5	5	5	0	5	5	5
CLASSIFICATIONS PARAMETERS											
NGI - Q											
RQD		98	90	90	90	90	90	90	90	90	90
Jn		0.5	1.0	3.0	5.0	5.0	3.0	5.0	1.0	1.0	1.0
Jr		1.5	1.5	1.2	1.2	1.2	1.2	1.0	1.0	1.0	1.0
Ja		1.0	3.0	4.0	4.0	5.0	2.5	4.0	3.0	3.0	5.0
Jw		1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
SRF		1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
CSIR - RMR											
RQD Rating		21.8	19.0	19.0	19.0	19.0	19.0	19.0	19.0	19.0	19.0
Discontinuity Spacing Rating		20.0	16.1	16.1	16.1	16.1	16.1	16.1	16.1	16.1	16.1
Joint Condition Rating		25.0	25.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0
Uniaxial Compressive Strength Rating		4.0	4.0	4.0	4.0	4.0	4.0	2.2	4.0	4.0	4.0
Groundwater Rating		10.0	10.0	10.0	10.0	10.0	10.0	15.0	10.0	10.0	10.0
Orientation Rating		0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
RSR											
Geology/Structure Rating		24	24	24	24	24	24	19	24	24	24
Discontinuity Spacing/Orientation Rating		36	30	30	30	30	30	30	30	30	30
Groundwater/Strength Rating		23	19	19	19	19	23	22	19	19	19
ROCK MASS RATINGS											
Q		294.9	45.0	9.0	5.4	4.3	14.4	4.5	30.0	30.0	30.0
RMR		81	74	54	54	54	54	57	54	54	54
RSR		83	73	73	73	73	77	71	73	73	73
SUPPORT PRESSURE CALCULATIONS											
Pressure from deflection	kPa	0.00	33.72	0.00	93.02	151.02	0.00	148.84	0.00	33.72	91.86
Pressure from Q	kPa	4.63	12.25	45.36	69.44	74.80	38.79	88.54	21.04	21.04	21.04
Pressure from RMR	kPa	21.40	28.97	51.33	51.33	58.49	51.33	47.76	51.33	51.33	51.33
BEDDING PLANE SHEAR STRENGTH											
Ø' (degrees)		56	27	17	17	16	26	16	18	18	18

LOCATION		P12	P13	P14	P6	P20	P5	P15	P3&P4	P2	P7
INPUT PARAMETERS											
Span	(metres)	4.3	4.3	4.3	4.3	4.3	7.6	4.3	4.3	5.5	4.3
Support Spacing	(metres)	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9
Beam deflection	(cm)	9	38	15	10	0	10	13	3	46	15
Load (calc. from deflection)	(kN/m)	440	1870	760	9	0	0	640	150	0	740
Roof Type (see Figure 5.1)		C	C	C	C	B	B	E	E	E	F
Discontinuity Spacing	(mm)	100	100	100	100	100	100	80	80	80	130
Uniaxial Compressive Strength	(MPa)	20	20	20	20	20	20	20	20	20	20
Groundwater Flow per 10m	(l/min)	0	0	2	0	0	0	5	5	5	0
CLASSIFICATIONS PARAMETERS											
NGI - Q											
RQD		56	56	56	56	56	56	44	44	44	68
Jn		5.0	5.0	5.0	5.0	5.0	5.0	5.0	2.0	3.0	5.0
Jr		1.0	1.0	2.0	1.0	1.0	1.0	0.9	0.9	0.8	0.8
Ja		2.5	2.5	2.5	2.5	2.5	2.5	4.0	3.0	3.0	3.0
Jw		1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
SRF		1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
CSIR - RMR											
RQD Rating		12.3	12.3	12.3	12.3	12.3	12.3	9.8	9.8	9.8	14.3
Discontinuity Spacing Rating		9.4	9.4	9.4	9.4	9.4	9.4	8.0	8.0	8.0	11.0
Joint Condition Rating		10.0	10.0	10.0	10.0	10.0	10.0	5.0	5.0	5.0	5.0
Uniaxial Compressive Strength Rating		2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2
Groundwater Rating		15.0	15.0	10.0	15.0	15.0	15.0	10.0	10.0	10.0	15.0
Orientation Rating		0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
RSR											
Geology/Structure Rating		19	19	19	19	19	19	19	19	19	19
Discontinuity Spacing/Orientation Rating		13	13	13	13	13	13	13	13	13	13
Groundwater/Strength Rating		22	22	19	22	22	22	15	15	15	18
ROCK MASS RATINGS											
Q		4.5	4.5	9.0	4.5	4.5	4.5	1.97	6.58	3.90	3.6
RMR		49	49	44	49	49	49	35	35	35	47
RSR		54	54	51	54	54	54	47	47	47	50
SUPPORT PRESSURE CALCULATIONS											
Pressure from deflection	kPa	102.33	436.88	172.09	0.00	0.00	0.00	148.84	34.88	0.00	172.09
Pressure from Q	kPa	88.43	88.43	35.09	88.43	88.43	88.43	129.45	54.81	89.91	118.89
Pressure from RMR	kPa	57.08	57.08	62.67	57.08	57.08	100.88	72.61	72.61	92.87	58.75
BEDDING PLANE SHEAR STRENGTH											
Ø' (degrees)		22	22	39	22	22	22	13	17	15	15

TABLE 5.5

OBSERVED CONDITIONS OF STUDY AREAS

AT LINGAN MINE

Location (Roof Type)	Roof Conditions	Groundwater Conditions	Remarks
L1 (G)	grey, fine to coarse grained siltstone with light grey, very fine to fine grained sandstone channels	generally dry	based on observations and data from bore-boles L10E-3A and L10E-2A, slabs very thick some fracturing at 0.1 m spacing slabs, 3-5 cm in immediate roof
L2 (H)	approx. 0.5 m of mudstone underlying very fine to fine grained, light to medium grey sandstone interbedded with med. to coarse grained, grey siltstone	generally dry	some fracturing at 0.1 m spacing slabs, 3-5 cm in immediate roof
L3 (C)	approx. 2 to 2.5 m of dark grey mudstone with plant fossils common underlying approx. 2 m of fine to coarse grained medium to dark grey siltstone underlying approx. 1 m of dark grey mudstone	generally dry	based on observations of roof conditions and data from borehole L10E-5, slabs fallen from roof 2-5 cm thick
L4 (G)	aprox. 4 m of very fine to fine grained light grey, slightly cacareous sandstone, channels within med. to coarse grained, dark grey siltstone	generally dry	based on observed roof conditions and data from borehole L10E-19.
L5 (F)	fine to coarse grained medium to dark grey siltstone with sandstone beds at approx. 2.5 m and 5.5 m from the roof with mudstone laminated throughout with a thin coal side seam at 0.6 m	generally dry	based on observed roof conditions and data from borehole L10E-19. L10E-24 roof slabbing 1-5 cm thickness

Location (Roof Type)	Roof Conditions	Groundwater Conditions	Remarks
L6 (F)	fine to coarse grained medium to dark grey siltstone with sandstone beds at approx. 2.5 m and 5.5 m from the roof with mudstone laminated throughout with a thin coal side seam at 0.6 m	generally dry	based on observed roof conditions and data from borehole L10E-19. L10E-24 roof slabbing 1-5 cm thickness
L7 (H)	med to coarse grained med to dark grey, siltstone interbedded with very fine grained, light grey sandstone with fine laminations and cross laminations	generally dry	roof slabbing 3-5 cm thickness based on observations and data from borehole L7W-2
L8 (H)	approx. 1.5 m of very fine grained, light to medium gray, slightly calcareous sandstone underlying approx. 10 m of fine to coarse grained med. to dark grey siltstone	generally dry	based on observations and data from bore-hole L7W-7 slabs fallen from walks 0.1-0.2 m thick
L9 (H)	interbedded very fine to fine grained, light grey sandstone and fine to med. grained, dark grey siltstone	generally dry	based on observed roof conditions and data from borehole L7W-9
L10 (H)	approx 3 m of med to coarse grained, med to dark grey siltstone with minor sandstone laminations underlying approx. 3 m of very fine to fine grained light grey sandstone with minor siltstone at 5.2 m	generally dry	based on observed roof conditions and data from borehole L7W-11, slabs fallen from roof 0.1-0.3 m thick
L11 (H)	dark grey siltstone interbedded with light grey sandstone	generally dry	

Table 5.6 - Rock Mass Classification of Lingan Mine

LOCATION		L3	L5	L6	L1	L4	L7	L8	L9	L10	L11	L2
INPUT PARAMETERS												
Span	(metres)	4	4.5	4.4	4.2	4.3	4.4	4.4	4.4	4.4	4.4	4.7
Support Spacing	(metres)	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9
Beam Deflection *	(cm)	8	25	8	5	25	33	0	36	0	25	20
Load (from deflection)	(kN/m)	86	270	86	54	270	356	0	576	0	400	216
Roof Type (see Figure 5.1)		C	F	F	G	G	H	H	H	H	H	H
Discontinuity Spacing	(mm)	100	100	100	100	200	100	100	100	200	200	100
Compressive Strength	(MPa)	21.0	32.0	32.0	64.0	64.0	49.0	49.0	49.0	49.0	49.0	42.0
Groundwater Flow per 10m	(l/min)	0	0	0	0	0	0	0	0	0	0	0
CLASSIFICATIONS PARAMETERS												
NGI - Q												
RQD		56	56	56	56	82	56	56	56	82	82	56
Jn		5.0	5.0	5.0	5.0	1.0	5.0	5.0	5.0	1.0	1.0	5.0
Jr		1.0	1.5	3.0	3.0	1.0	1.5	3.0	2.0	3.0	2.0	1.5
Ja		3.5	6.0	7.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	4.0
Jw		1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
SRF		1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
CSIR - RMR												
RQD Rating		12	12	12	12	17	12	12	12	17	17	12
Discontinuity Spacing Rating		9	9	9	9	14	9	9	9	14	14	9
Joint Condition Rating		5	5	5	20	20	20	20	20	20	20	5
Uniaxial Compressive Strength Rating		2	3	3	5	5	4	4	4	4	4	4
Groundwater Rating		15	15	15	15	15	15	15	15	15	15	15
Orientation Rating		0	0	0	0	0	0	0	0	0	0	0
RSR												
Geology/Structure Rating		24	24	24	24	24	24	24	24	24	24	24
Discontinuity Spacing/Orientation Rating		13	13	13	13	23	13	13	13	23	23	13
Groundwater/Strength Rating		12	12	12	18	22	12	12	12	18	18	12
ROCK MASS RATINGS												
Q		3.2	2.8	4.8	11.3	27.2	5.6	11.3	7.5	81.6	54.4	4.2
RMR		44	45	45	62	70	61	61	61	70	70	45
RSR		49	49	49	55	69	49	49	49	65	65	49
SUPPORT PRESSURE CALCULATIONS												
Pressure from Deflection	kPa	21.50	60.00	19.55	12.86	62.79	80.91	0.00	130.91	0.00	90.91	45.96
Pressure from Q	kPa	98.93	68.95	28.81	21.72	21.74	54.73	21.72	37.29	5.02	8.63	60.24
Pressure from RMR	kPa	58.21	64.54	63.11	41.77	33.02	44.77	44.77	44.77	34.81	34.81	66.65
BEDDING PLANE SHEAR STRENGTH												
Θ' (degrees)		16.0	14.0	23.2	45.0	18.4	26.6	45.0	33.7	45.0	33.7	20.6

TABLE 5.7

OBSERVED CONDITIONS OF STUDY AREAS AT PHALEN MINE

Location (Roof Type)	Roof Conditions	Groundwater Conditions	Remarks
PH1 (H)	approx 1 m of grey massive mudstone with some listric surfaces; slabs fallen from roof approx. 100 mm thick; underlying a siltstone unit	generally dry	
PH2 (H)	approx 1 m of grey massive mudstone with some listric surfaces; slabs fallen from roof approx. 100 mm thick; underlying a siltstone unit	generally dry	
PH3 (H)	approx 1 m of grey massive mudstone with some listric surfaces; slabs fallen from roof approx. 100 mm thick; underlying a siltstone unit	generally dry	
PH4 (H)	approx. 0.5 m of grey mudstone with some listric surfaces underlying a siltstone unit	generally dry	
PH5 (E)	grey laminated mudstone with two 0.2 m thick coal seams at 0.3 m and 0.45 m from the roof	generally dry	
PH6 (E)	grey laminated mudstone with two 0.2 m thick coal seams at 0.3 m and 0.45 m from the roof	generally dry	some slabs fallen from roof approx. 100-300 mm thick
PH7 (E)	grey mudstone with 2 coal side seams .05 m and 0.2 m thick at a height of 0.1 m and 0.5 m from the roof	generally dry	

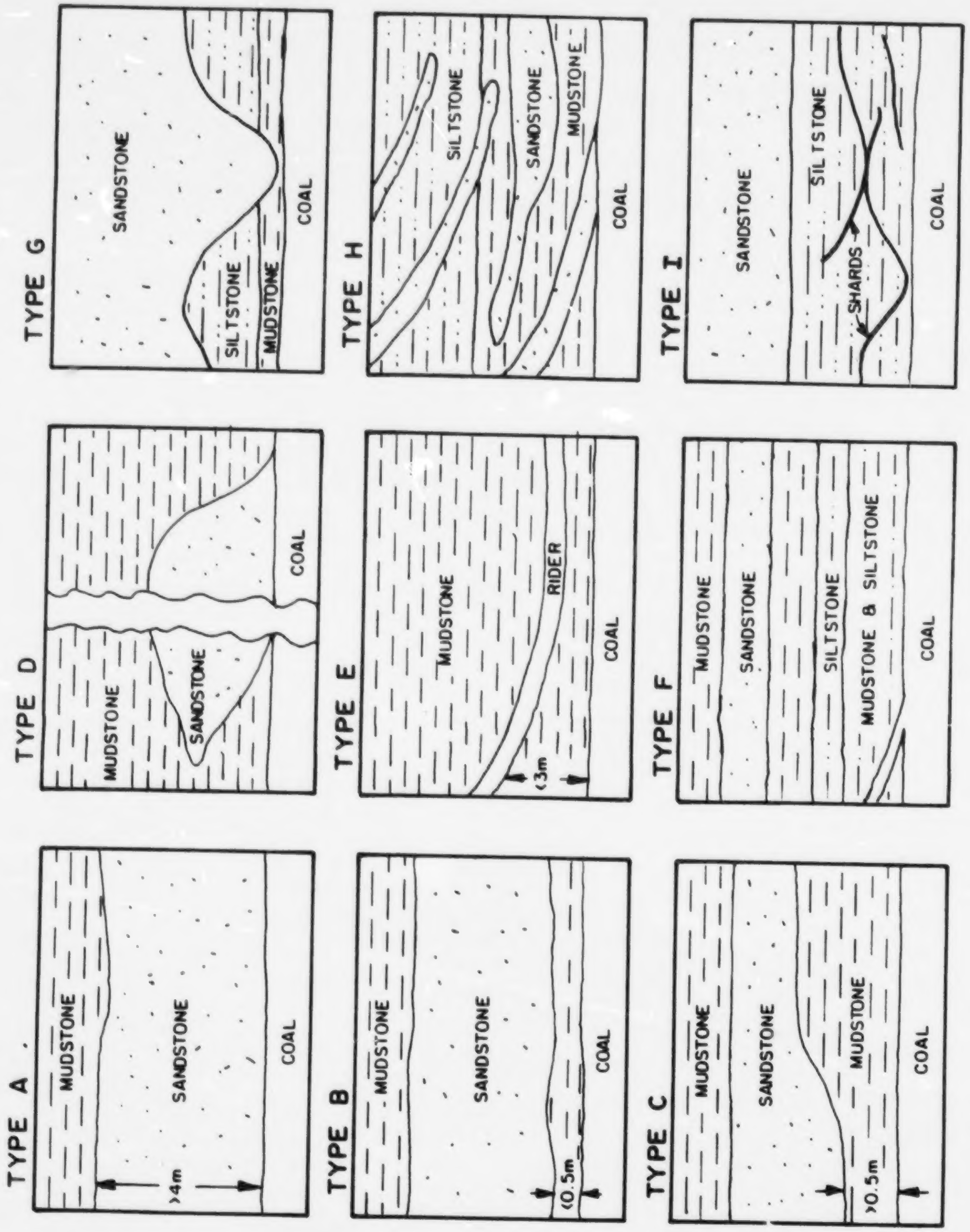
Location (Roof Type)	Roof Conditions	Groundwater Conditions	Remarks
PH8 (A)	approx. >2 m of fine to medium grained sandstone	generally dry	
PH9 (B)	approx. 1 m of grey mudstone underlying a siltstone unit	generally dry	
PH10 (E)	very weak, grey laminated mudstone with coaly bands and a coal side seam approx. 0.8 m thick at 0.3 m from the roof	generally dry	
PH11 (F)	approx. 0.5 m of weak, laminated mudstone underlying interbedded siltstone/sandstone units	generally dry	
PH12 (G)	interbedded mudstone and siltstone with sandstone channel	generally dry	
PH13 (A)	approx. >3 m thick fine to med grained sandstone unit	generally dry	
PH14 (G)	grey mudstone with fine to med grained sandstone channel within 0.2 m of the roof	generally dry	
PH15 (G)	grey mudstone with fine to med grained sandstone channels	generally dry	
PH16 (A)	fine to med grained sandstone unit with plant fossils, approx. >2 m thick	generally dry	
PH17 (F)	approx. 0.3 m of grey mudstone underlying approx. 1.2 m of siltstone underlying >1 m of sandstone	generally dry	

Table 5.8 - Rock Mass Classification for Phalen Mine

LOCATION		PH8	PH13	PH16	PH15	PH9	PH14	PH7	PH6	PH5	PH4	PH3	PH10	PH11	PH17	PH12	PH2	PH1
INPUT PARAMETERS																		
Span	(metres)	3.7	3.7	3.5	3.5	3.7	3.7	3.5	3.5	3.5	3.5	3.5	3.7	3.7	3.5	3.7	3.5	3.5
Support Spacing	(metres)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Beam Deflection	(cm)	0	0	0	5	0	5	0	0	3	3	8	0	0	0	0	0	25
Load (calc. from deflection)	(kN/m)	0	0	0	50	0	50	0	0	30	30	80	0	0	0	0	0	300
Roof Type (see Figure 5.1)		A	A	A	B	C	D	E	E	E	E	E	E	F	F	G	H	H
Discontinuity Spacing	(mm)	600	600	500	100	100	100	100	200	200	300	100	80	80	100	100	100	100
Compressive Strength	(MPa)	58	58	58	58	28	46	28	28	28	28	28	28	46	46	46	40	40
Groundwater Flow per 10m	(l/min)	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
CLASSIFICATIONS PARAMETERS																		
NGI - Q																		
RQD		98	98	97	56	56	56	56	82	82	90	56	44	44	56	56	56	56
Jn		0.5	0.5	1.0	5.0	5.0	5.0	5.0	1.0	1.0	1.0	5.0	10.0	10.0	5.0	5.0	5.0	5.0
Jr		3.0	3.0	1.5	2.0	0.9	2.0	1.5	1.2	1.0	1.0	0.9	1.5	1.5	0.9	2.0	1.0	1.0
Ja		1.2	1.2	1.2	4.0	3.5	4.0	5.5	4.0	4.0	3.5	3.5	7.0	4.0	3.5	5.0	3.5	3.5
Jw		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
SRF		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
CSIR - RMR																		
RQD Rating		22	22	21	12	12	12	12	17	17	19	12	10	10	12	12	12	12
Discontinuity Spacing Rating		20	20	19	9	9	9	9	14	14	16	9	8	8	9	9	9	9
Joint Condition Rating		30	30	25	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Uniaxial Compressive Strength Rating		5	5	5	5	3	4	3	3	3	3	3	3	4	4	4	4	4
Groundwater Rating		15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15
Orientation Rating		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
RSR																		
Geology/Structure Rating		24	24	24	24	24	24	24	24	24	26	24	24	26	24	24	24	24
Discontinuity Spacing/Orientation Rating		30	30	30	13	13	13	13	23	23	30	13	13	13	13	13	13	13
Groundwater/Strength Rating		22	22	22	18	12	12	12	18	18	18	12	12	12	12	12	12	12
ROCK MASS RATINGS																		
Q		491.6	491.6	120.8	5.6	2.9	5.6	3.1	24.5	20.4	25.7	2.9	0.9	1.6	2.9	4.5	3.2	3.2
RMR		91	91	85	46	45	46	45	53	53	58	45	41	42	46	46	45	45
RSR		76	76	76	55	49	49	49	65	65	72	49	49	49	49	49	49	49
SUPPORT PRESSURE CALCULATIONS																		
Pressure from deflection	kPa	0.0	0.0	0.0	14.3	0.0	13.5	0.0	0.0	8.6	8.6	22.9	0.0	0.0	0.0	0.0	0.0	85.7
Pressure from Q	kPa	2.0	2.0	8.8	41.0	113.8	41.0	67.0	18.8	23.9	22.1	113.8	140.7	116.7	113.8	44.2	98.9	98.9
Pressure from RMR	kPa	8.2	8.2	13.6	48.8	53.3	52.2	50.4	42.5	42.5	38.3	50.4	57.1	56.0	49.4	52.2	49.7	49.7
BEDDING PLANE SHEAR STRENGTH Ø' (degrees)		68	68	51	27	14	27	15	17	14	16	14	12	21	14	22	16	16

TYPICAL ROOF TYPES FOR THE SYDNEY COALFIELD

FIGURE 5.1


TYPE A
MUDSTONE
SANDSTONE
COAL
>4m
TYPE B
MUDSTONE
SANDSTONE
COAL
<0.5m
TYPE C
MUDSTONE
SANDSTONE
MUDSTONE
COAL
>0.5m
TYPE D
MUDSTONE
SANDSTONE
COAL
TYPE E
MUDSTONE
RIDER
COAL
<3m
TYPE F
MUDSTONE
SANDSTONE
SILTSTONE
MUDSTONE & SILTSTONE
COAL
TYPE G
SANDSTONE
SILTSTONE
MUDSTONE
COAL
TYPE H
SILTSTONE
SANDSTONE
MUDSTONE
COAL
TYPE I
SANDSTONE
SILTSTONE
SHARDS
COAL

REFERENCE: CAPE BRETON DEVELOPMENT CORPORATION
GEOLOGICAL RECORDS

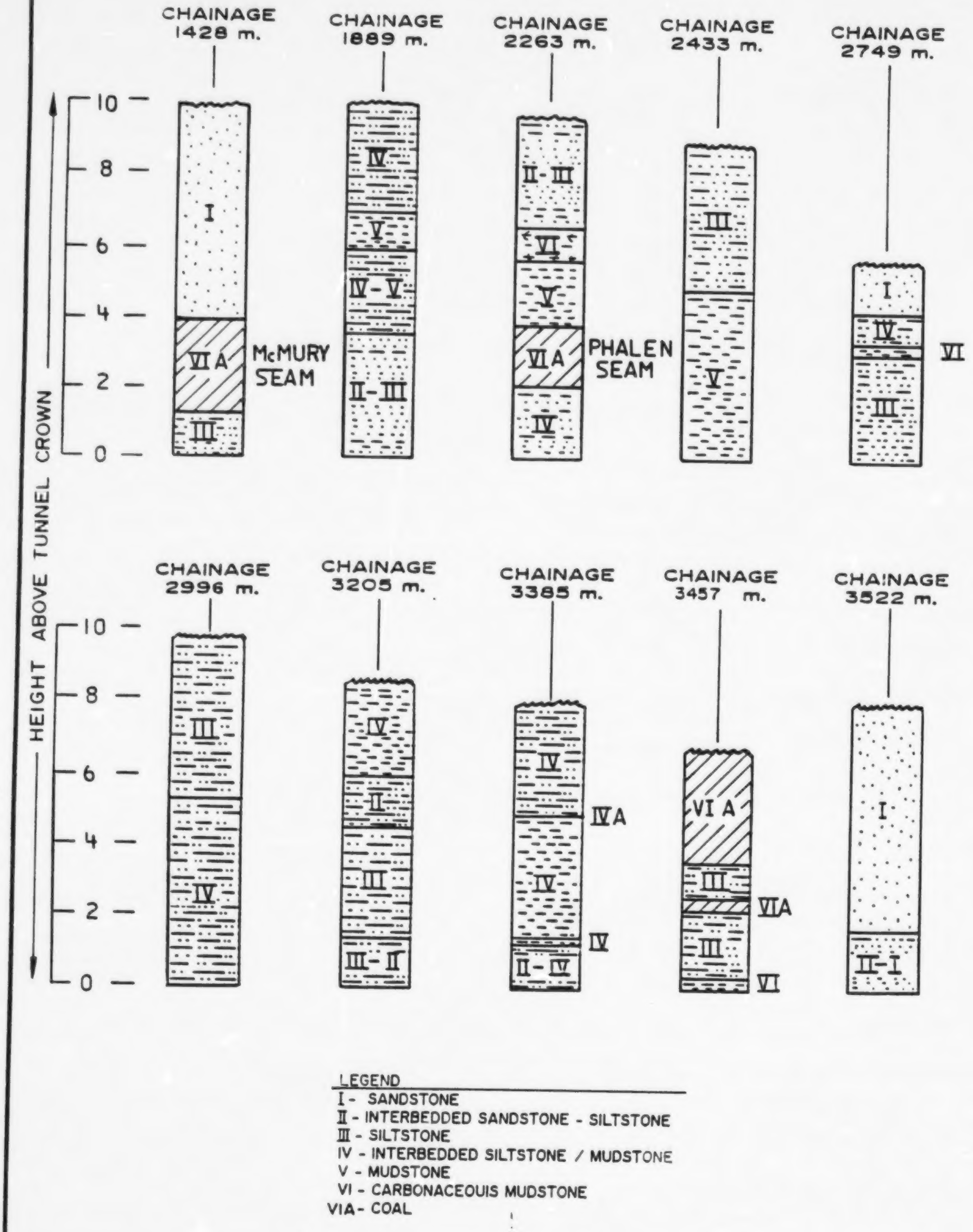
Date MARCH, 1989
Project 881-1719

Golder Associates

Drawn CPAM
Chkd

STRATIGRAPHY FOR DONKIN TUNNEL STUDY AREAS

FIGURE 5.2



CHAINAGE 1428 m.
CHAINAGE 1889 m.
CHAINAGE 2263 m.
CHAINAGE 2433 m.
CHAINAGE 2749 m.
HEIGHT ABOVE TUNNEL CROWN
10
8
6
4
2
0
I
VI A
McMURY SEAM
III
IV
V
IV - V
II - III
II - III
VI
V
VI A
PHALEN SEAM
IV
III
V
I
IV
VI
III
CHAINAGE 2996 m.
CHAINAGE 3205 m.
CHAINAGE 3385 m.
CHAINAGE 3457 m.
CHAINAGE 3522 m.
III
IV
IV
II
III
III - II
IV
IV A
IV
IV
II - IV
VI A
III
VI A
III
VI
I
II - I

LEGEND

I - SANDSTONE
II - INTERBEDDED SANDSTONE - SILTSTONE
III - SILTSTONE
IV - INTERBEDDED SILTSTONE / MUDSTONE
V - MUDSTONE
VI - CARBONACEOUIS MUDSTONE
VIA - COAL

NOT DRAWN TO SCALE

Date MAY 11, 1989
Project 891 - 1719

Golder Associates

Drawn D. M.
Chkd. CEJ

6.0 DISCUSSIONS OF RESULTS

6.1 Modifications to the Classification Systems

Three commonly used classification schemes have been evaluated for their effectiveness for classifying the rock mass conditions at four operating mines in the Sydney Coalfield. For each operating mine, initial tests using the conventional published ratings were found to be of limited success. This is considered to have arisen mainly because the ratings used for describing the rock mass conditions were based on those for describing hard rock situations. Much more success has been achieved with modifications to the rock mass descriptors based partly on those originally recommended by the Q, RMR and RSR systems, and partly on the modified ratings established specifically from this study for coal measure rocks.

The rock mass classification indices resulting from the recent evaluation of the conditions in the coal mines and the support pressures back-calculated from on-site observations of beam deflections (for Prince, Lingan and Phalen Mines) and those from empirical relationships have been presented in Tables 4.1, 5.4, 5.6 and 5.8 for Donkin, Prince, Lingan and Phalen Mines respectively. The development of the modified classification descriptors is summarized below.

As evidenced in earlier discussions, one of the main difficulties of applying the three studied rock mass classifications in their original published form was encountered in applying some of the original Q and CSIR-RMR rock mass descriptors to coal measure rocks. For the Q system, the descriptors for the J_n and J_a parameters required modification in order to reflect the bedded nature of discontinuities and their surface conditions which are more relevant to those encountered in coal mining environments (see Table 4.3).

In the Sydney Coalfield, very few joints occur that exert any significant influence on rock mass behaviour at depth in the mines. In almost all situations observed underground the predominant discontinuities are the bedding surfaces and contacts between different rock units. Hence, the modified descriptors for the J_n parameter relate to bed thicknesses; the thicker the beds the higher the J_n rating. The range of the rating for the J_n parameter, however, has purposely been kept similar to Barton's original range of ratings in order to maintain the final Q values consistent with anticipated support requirements.

The descriptors for the J_a parameters have also been modified to reflect the condition of the bedding surfaces and contacts between different coal measures type rock units. Again, in order to maintain the original intent of the Q classification system, the range of the rating for the joint alteration descriptors (J_a) has been deliberately set to encompass the same spread as that in the original Q system for hard rock discontinuity conditions but the categorization has also been modified (see Table 4.3).

For the Geomechanics Classification, RMR system, the descriptors for the joint condition parameter also required modification in order to relate them to the condition of contacts between various coal measure strata. However, the range of rating for the modified descriptors has again been held similar to the original range of ratings suggested by Bieniawski (1973) in order that final RMR values are representative of observed rock conditions.

No modifications were undertaken of the ratings for the RSR system although the terminology for the descriptors covering discontinuity spacing have been re-written in terms of bed separations, rather than as discontinuity spacing (see Table 4.5).

6.2 Application of the Classification Systems to the Sydney Coalfield

By establishing the appropriate J_n and J_a descriptor ranges for the Q system, and the joint condition terminology descriptors for the RMR classification system, the application of these classification schemes to assessing the roof conditions encountered in the Sydney coal mines could be rationalized. The rating for each of the parameters was assigned in accordance with the details of the roof condition at the study areas. However, it may be noted from the results presented in Section 5 that even though the roof condition at two locations are grouped under the same rock type, different classification indices may result due to local variations in stratigraphy or in the slight difference in parameter descriptors of this classifications. This is not uncommon, even in hard rock situations from which the classifications were originally derived. In fact, in practice, it has been found most efficient to undertake Q and RMR classifications independently then average the results in order to define the "best estimate" value for the rock mass classification rating. For some hard rock situations, typically where rock quality is good and the rock is relatively massive, the spread between the quality assessed by each system is negligible, but for schistose rocks, it is often very wide. This same dilemma of different values of rock quality assessed according to the different classification systems is still evident in the range of classification indices derived for the different roof types in the four coal mines involved in this study. The quality indices determined from each of the classifications for the various different rock conditions in the Sydney Coalfield are summarized below:

Roof Type	Q	RMR	RSR
A	45.0 - 491.6	74 - 91	76 - 83
B	4.3 - 30	46 - 57	55 - 77
C	1.3 - 9.0	44 - 57	49 - 74
D	4.5 - 5.6	46 - 49	49 - 54
E	0.9 - 25.7	35 - 58	49 - 72
F	1.3 - 17.5	42 - 82	49 - 93
G	4.5 - 27.2	46 - 70	49 - 69
H	3.2 - 81.6	45 - 70	49 - 65
I	-	-	-

Roof types A and B (see Figure 5.1 for definitions) represent good roof conditions. Roof types C to H, because of variations in the competence of the rock types forming the interbedded units, comprise a range of rock mass quality. Roof type I was not encountered in the study areas examined.

Based on the Q and RMR rock mass quality indices, roof conditions within the Sydney coalfield mines range from "very poor" to "exceptionally good" according to the Q system and from "poor" to "very good" based on the RMR system. These groupings are plotted in Figure 6.1 as a correlation graph between the Q and RMR systems. As is evident, the correlation between the two systems is good. In general, although it is rather difficult to generalize on rock mass quality purely based on a broad category of rock types, the results are encouraging and suggest that rigorous use of such classifications is possible provided that detailed information on the stratigraphy at the roof is incorporated in any approach to rationally evaluating rock mass quality in the mines.

In applying the classification systems to the coalfield rocks, several points worthy of further consideration were noted:

a. In the Q system, the Q index is strongly influenced by the J_n value. Other factors being equal, a massive unit would be ascribed a significantly higher (about 10 times) Q index than a thinly bedded layer. Although consistent with the generally observed stability conditions of openings in bedded strata, this can lead to low Q values (poor quality rocks) and hence extremely high support pressure predictions if the characteristic spacing of beds is in the millimetre range. Therefore, some limiting, geometrically controlled factor dependent on tunnel span for instance, needs incorporation into the expression for support prediction from the Q system in order to avoid this problem.

b. The Q system can provide a good measure of the geomechanical strength characteristics of individual bedding planes as the values of $\tan^{-1}(J_r/J_a)$ should generally be approximately equal to the friction angle (ϕ') appropriate for the discontinuity surfaces. This fact thus provides a ready check on the accuracy of the parameter input data, particularly as the computed ϕ' values should always realistically reflect assessed bedding plane shear strength values (refer to bottom lines on tables 5.4, 5.6 and 5.8).

c. For the RMR system, the discontinuity surface condition rating is very sensitive to interpretation of controlling bed thicknesses; i.e. like J_n, it can have a marked effect on the overall RMR value calculated. Based on the overall correlation of the RMR and Q values (see discussions in Section 6.3), the ratings for discontinuity surface conditions established by back-analysis of the Donkin data appear reasonable. However some further refinement of the codings may still be warranted.

d. The rating adjustment for discontinuity orientations in the RMR systems is difficult to apply and often the application is arbitrary. It was not used in this study as the excavations are basically parallel to the main bedding structure and there are very few cross joints present in the rock mass. In fact, this parameter can be applied in a similar fashion to the SRF of the Q system which is related to excavation geometry and characteristics, more than to the rock quality. The characteristics of the bedding and the interlayering effects of the strata are in general, more appropriately covered by other rating factors related to bedding plane spacing and discontinuity surface conditions.

e. Among the three classification schemes, the RSR system is the only one which explicitly adjusts for the use of a tunnel boring machine for excavation. Again, this adjustment factor is similar in application to the SRF of the Q-system, or the orientation rating for the RMR system, and therefore can be considered solely excavation dependent. Adjustment factors for different mining conditions, i.e. to assess the effects of advancing longwall faces on gate road conditions can be treated similarly, much along the lines of the Laubscher modification to the CSIR-RMR system for hard rock mining usage Laubscher (1976).

Based on an assessment of the Lingan deflection data as being representative of the influence of advancing longwall extraction and gob development on support pressures, the following tentative Q system stress reduction factors (SRF) and RMR system modifiers are suggested:

	SRF	RMR Derating
Low modulus rock units*	10	-20
High modulus (brittle fracturing) rock units **	30	-30

* mudstones, siltstones, generally of relatively low Q or RMR based on initial assessment, without modification

** sandstones, interbedded sandstones/siltstones generally of relatively high Q or RMR based on initial assessment, without modifications.

f. The modified Q, RMR and RSR classification indices developed as part of this study were derived within the general framework of the original systems. The main effect of these proposed modifications is that by categorization of the descriptors of some of the parameters to reflect changes in the character and spacing of bedding planes, rather than jointing, rationalization of the original ratings can be achieved. This allows definitive and reliable use of the systems in sedimentary rock strata.

g. The categorization of the discontinuity parameter descriptors for the Q and CSIR-RMR classification systems that have been developed in this study enable a straightforward application of these systems to the rock types and mine openings of the Sydney Coalfield. The modified descriptors directly address the relevant conditions which control the stability of the mine openings. The RSR system can also be rationally used, however, it is much less sensitive and much more general than the other two systems. The only necessary modifications for its use in coal measure rocks require the joint spacing descriptor for flatly dipping joints to be related to the bedding plane spacing and the joint condition parameter to be modified from describing the weathering state to assessing the characteristics of inter-unit contacts.

In conclusion, each of the three classification schemes included in this study, as modified in accordance with tables 4.2, 4.4 and 4.5 can be applied with confidence to the description of the Sydney Coalfield rocks. The Q and RMR systems, however, provide better definition of rock mass conditions and hence of potential support requirements. If the RSR system is to be used in the coalfield, input parameters for this system should be based on the Q and RMR parameter values based on the correlations presented in this study.

The effect of advancing longwall on the rock mass characteristics can be taken into account by derating the Q and RMR indices as previously discussed.

6.3 Correlation between the Rock Mass Classification Indices

Several correlations between RMR and Q values have been provided over the years. Probably the best known is that proposed by Bieniawski (1973). A total of more than 100 case histories from the Scandinavian countries, South Africa, United States, Canada, Australia and Europe were analysed and the following relationship was developed:

$$RMR = 9 \ln Q + 44 \qquad (6.1)$$

The data points used by Bieniawski and the line representing this best fit equation 6.1 is plotted in Figure 6.1. The band which contains 90% of the data points is also shown as are data points obtain from the rock mass classifications undertaken for the four mines examined. It is readily evident that the relationship defined by

equation 6.1 also appears to be reasonably representative for correlations between RMR and Q values for the Sydney Coalfield data. In fact, it may also be noted that over 90% of all the data points from this study also fall within the same band (based on Bieniawski's data).
The correlation between the RSR and Q values was also examined by plotting the RSR values against ln Q as shown in Figure 6.2. It is interesting to note that the same form of relationship as used between the Q and RMR values:

$$RSR = 9 \ln Q + 44 \qquad (6.2)$$

still appears to be applicable.

Based on data from New Zealand, the correlation between the Q values and both the RMR and RSR system indices determined by Rutledge and Preston (1978) are as follows:

$$\begin{aligned} RMR &= 13.5 \log Q + 43 \\ &= 5.9 \ln Q + 43 \qquad (6.3) \end{aligned}$$

and

$$\begin{aligned} RSR &= 13.3 \log Q + 46.5 \\ &= 5.8 \ln Q + 46.5 \qquad (6.4) \end{aligned}$$

There is considerable similarity between these equations for the RMR vs Q (eqn 6.3) and RSR vs Q (eqn 6.4) relationships to those presented by Bieniawski for RMR vs Q (eqn. 6.1) and that for RSR vs Q (eqn 6.2) determined in this study. In order to check that the values computed from application of the RSR and RMR systems to classification of coal measure rocks was still consistent with the original intent of the classifications, an attempt was made to cross-check Sydney coalfield data against the relationship of the RMR and RSR values determined by Rutledge and Preston (1978).

$$RSR = 0.77\ RMR + 12.4 \qquad (6.5)$$

As illustrated by the data presented in Figure 6.3, the relationship (eqn. 6.5) still appears to be applicable in correlating the data from this study. Almost all of the data falls within the band which brackets 90% of all the plotted chart data.

In summary, the correlations between the three classification indices when applied to the coal measures rocks examined in this study (using the modified descriptor codes) are consistent with those determined by others based on different sets of data for different rock types.

6.4 Validity of Support Pressures Assessment

For both the Q and RMR systems, empirical support pressures can be evaluated directly from the classification indexes and the related parameters as follows:

Q System (Barton et al 1973)

$$\text{support pressure} \quad p = \frac{2\ J_n^{1/2} (Q)^{-1/3}}{3\ J_r} \qquad (\text{kg/cm}^2) \qquad (6.6)$$

RMR System (Unal 1983)

$$\text{support pressure} \quad p = \left(\frac{100 - RMR}{100}\right) \gamma.B. \qquad (\text{kPa}) \qquad (6.7)$$

where:
Q and RMR are classification indices
J_n, J_r are discontinuity condition parameters for the Q system
γ is the density of the roof rock (kN/m^3)
B is the span of the opening (m)

In evaluating support pressures by these expressions, it should be realized that there are some fundamental differences in the way that these expressions were developed.

For the Q system, the support pressure is independent of the excavation size. It is related solely to the Q values and the factors reflecting discontinuity roughness and spacing. In consequence, the weaker the rock the higher the pressure. In fact, for very poor rock conditions ($Q << 1$), the predicted support pressure can become in some cases unreasonably high (it generally exceeds 250 kPa for some of the cases examined) and would increase further for lower Q values.

On the other hand, the empirical support pressure equation developed for use with the RMR system was derived from roof falls observed in coal mines. Hence the only variables in the expression (equation 6.6) are the RMR and the span of the opening. It may also be noted from this equation that the maximum pressure occurs when the RMR value is zero and the limiting value is equal to $B.\gamma$. This suggests that the maximum height of loosened rock above the opening would be approximately equal to the span under such limiting conditions. It is therefore essential that some limiting relationship be developed for the Q system to bound the realm of real and inconceivable support pressure values.

In the derivation of the various parameter ratings for the Q and RMR systems, the empirical support pressures derived from equations 6.6 and 6.7 were used to match the pressures measured by the load cells or estimated from the displacement measurements at the Donkin Mine. Obviously, as long as the Q values are within the central range of applicability of equations 6.6 and 6.7, the results are valid, as shown in Table 4.1. This indicates that the support pressures derived from the Q and RMR systems are in reasonable agreement provided that the rock qualities fall within the range encountered at Donkin.

However, as discussed in Sections 5.2 to 5.4, some discrepancies between the support pressures derived from the two systems become apparent when the support pressures were calculated for poorer rock conditions or for less well controlled support practices as evidenced in the other mines as summarized below:

a. For high rock mass quality (fair to very good according to the Q-system), the support pressures calculated from eqn 6.6 using the Q-system ratings are generally somewhat lower than the corresponding pressures obtained from eqn 6.7 using the RMR values. This may in fact be realistic as the RMR based equation was initially developed from observation of roof falls in rocks not of the same quality range.

b. For very low rock mass quality, (poor to very poor according to the Q system), the pressure predicted by the Q equation becomes increasingly less realistic especially in the case of Q values significantly less than 1. This points to the need to incorporate a span/geometry related upper bound control into the support pressure equation 6.6.

c. The support pressures estimated from the beam deflections in the various drivages do not always match those expected based on observed conditions. In fact there is no consistent correlation between this set of values and those determined from the rock mass classifications. This, however, is mainly attributed to the wide range in quality of the support installation practices in the various mines. A summary of the observed support conditions is given in Table 6.1.

The ranges of support pressures estimated from the Q and RMR classification systems, and the rock displacement/beam deflections are summarized as follows:

		Range of Support Pressure (kPa) [loosened height (m)] from		
Mine	Range of Rock Type	Deflection	Q	RMR
Donkin	C,E,F	39-104 [1.5-4]*	40- 86 [1.5-3.3]	36-99 [1.4-3.5]
Prince	A,B,C,D,E F	0-435 [0-16.7]**	5-130 [0.2-5]	21-101 [0.8-3.9]
Lingan	C,F,G,H	0-131 [0-5]**	5- 99 [0.2-3.8]	33-67 [1.3-2.6]
Phalen	A,B,C,D,E F,G,H	0-86 [0-3.3]**	2-141 [0.1-5.4]	8-57 [0.3-2.2]

*Based on rock displacement ** Based on beam deflection

It may be noted that in general the loosened rock heights back-calculated from the roof beam deflection data appear to be within the same range estimated from the RMR values. For poor to very poor rock quality as defined by the Q-system, the estimated pressures and the computed maximum height of loosened rock are inconsistent with observed behaviour. This again points to the need for use of a geometrical, upper bound cut-off for the original Barton Q-system support pressure equation.

Due to the lack of displacement records in the mines with the exception of Donkin, it is difficult at this stage to delineate, in the other mines, the loosened zone which results primarily from a combination of loosening due to the passive nature of the support and rock quality. The extensometer installed at Phalen Mine indicates an active zone of displacement of about 2 m. This seems to fall within the range of prediction from both the RMR and Q systems, provided that the rock mass quality is not so poor that the original Q system equation (6.6) gives unrealistically high estimations.

In summary, it is considered that while the empirical support pressures relationships are useful in estimating support pressures from the classification values, further detailed correlation work needs to be undertaken at each of the mines in order to ascertain that projection of the experience from one area of the mine to another is consistent and valid. Further, there is a definite need to establish a detailed data base from which to define a maximum height to span geometry term for incorporation into the support pressure prediction equation for the Q system. Until such time as an appropriate geometry term is developed for full reliable prediction of support pressures from the Q system, the support pressures estimated from the original empirical relationship of equation 6.6 (for the Prince, Lingan and Phalen Mines given in the Q and RMR systems) should be treated in relative terms and only as a general guide for assessing support requirements. Correlation of both rock mass classification indices with physically measured and observed performance of the support would aid considerably in projecting the experience in one area of a mine to other areas on a quantitative basis.

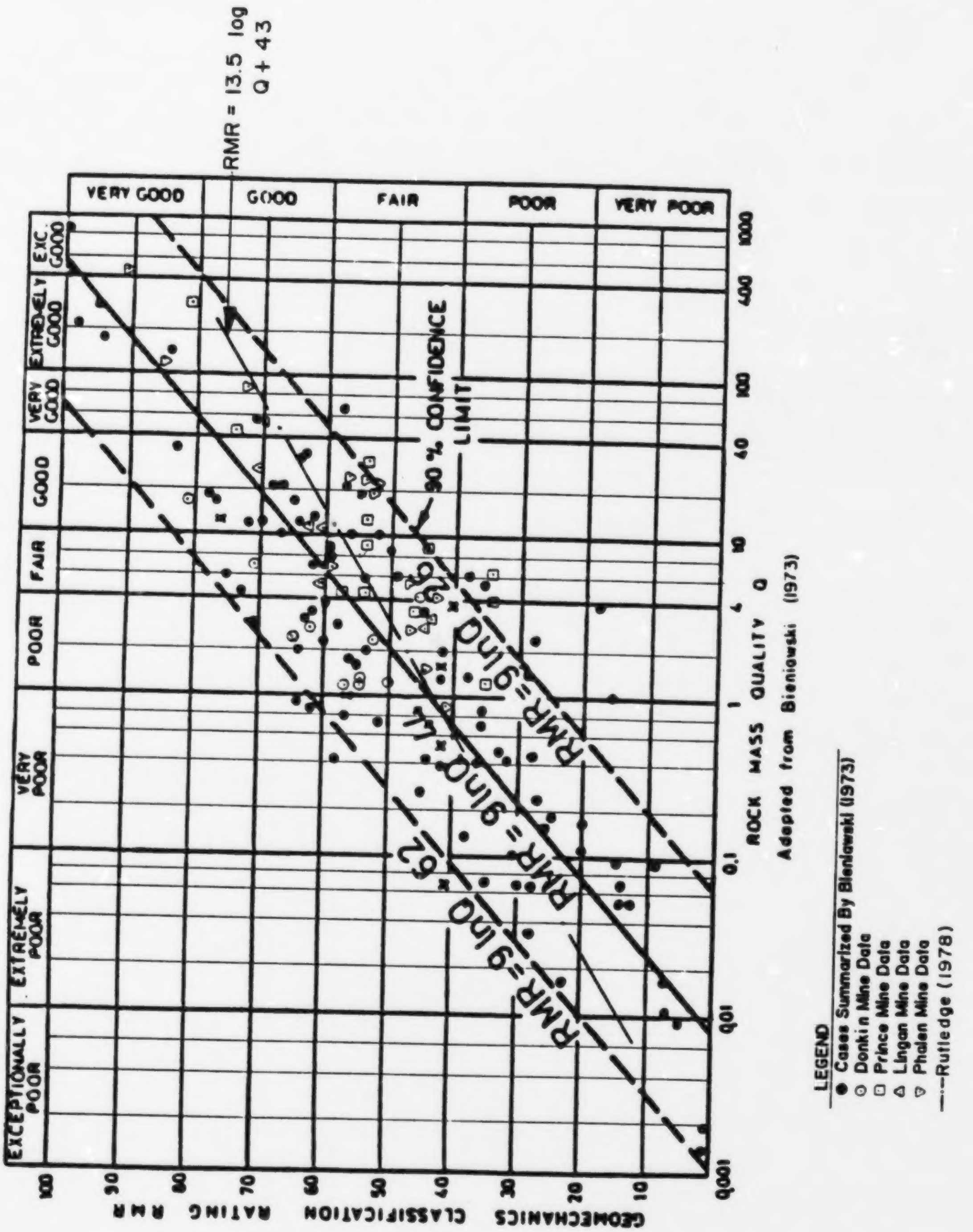
TABLE 6.1

OBSERVED SUPPORT CONDITIONS OF STUDY AREAS IN SYDNEY COALFIELD MINES

MINE	SUPPORT CONDITIONS
Donkin	- 150 x 150 mm ring beams generally tight against roof and side walls - rock fragments at crown supported by mesh panels between ring beams - no loose areas below 120° arch from crown
Prince	- square steel sets (4.27 m, 4.87 m or 5.5 m long roof beams) - back of opening may be tight against support or the gap may be filled with wood blocking - approximately 0.1 to 0.5 gap between the sidewalk and leg support
Lingan	- rigid and yielding steel arches - generally 0.2 - 0.5 m gap between arch support and rock all the way around with wood blocking in between - legs of arches are not always vertical - may be affected by installation
Phalen	- asymmetrical Hollybank steel arch - generally 0.3 m to 0.5 m gap between sidewalls and support legs and 0.1 m to 0.3 m gaps between back and roof beam - gaps usually filled with wood blocking

CORRELATION BETWEEN GEOMECHANICS CLASSIFICATION AND Q SYSTEM

FIGURE 6.1



Adapted from Bieniawski (1973)

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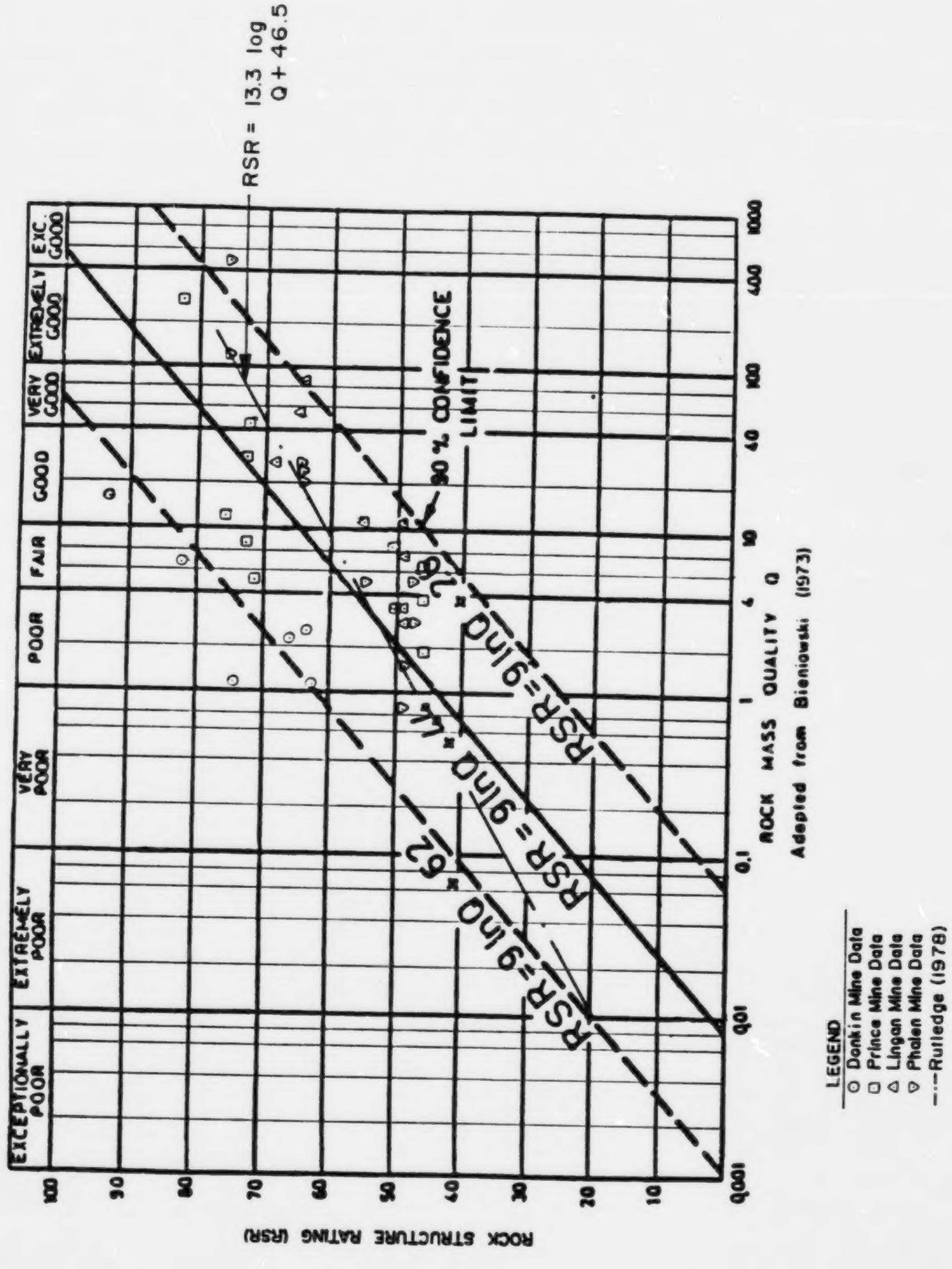
CORRELATION BETWEEN RSR CLASSIFICATION AND Q SYSTEM

FIGURE **6.2**



Adapted from Bieniawski (1973)

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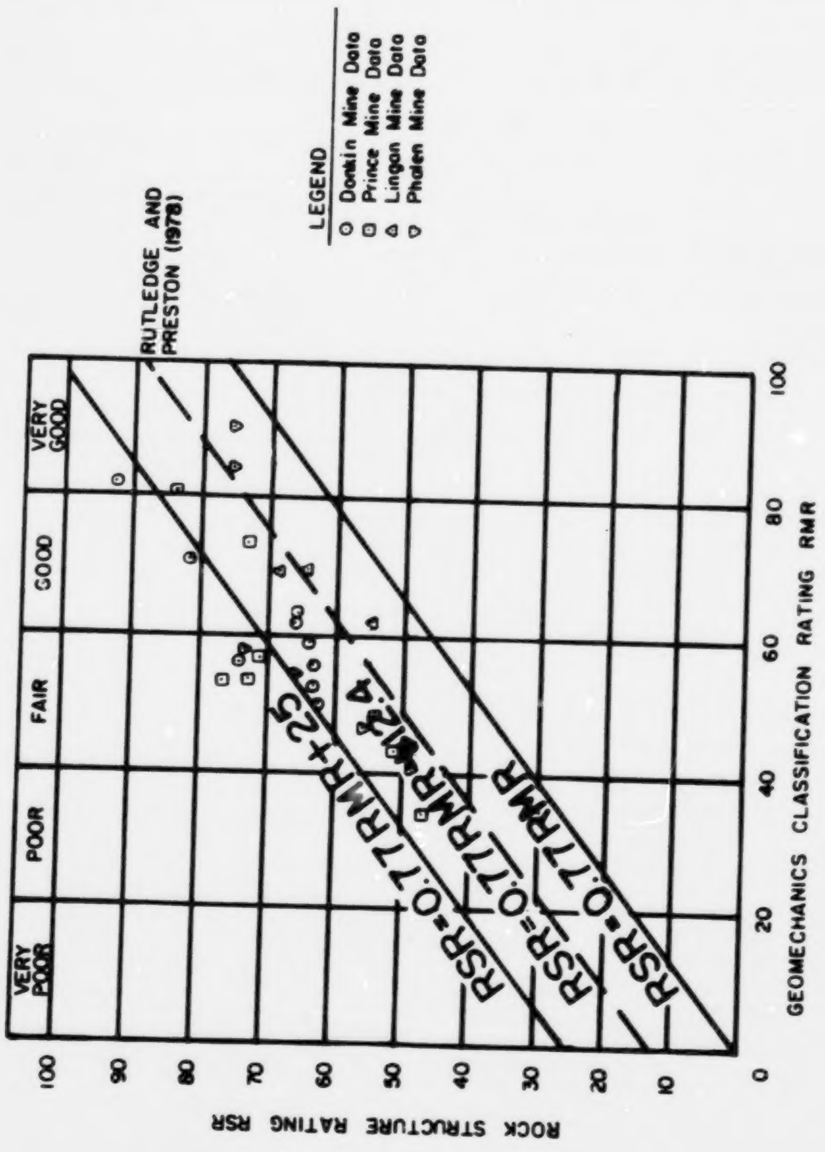
CORRELATION BETWEEN RSR AND RMR CLASSIFICATIONS

FIGURE **6.3**



Date **MARCH, 1989**
Project **881-1719**

Golder Associates

Drawn **CPAM**
Chkd

7.0 CONCLUSIONS AND RECOMMENDATIONS

Three rock mass classifications, the Norwegian Geotechnical Institute (NGI) Q system, the Geomechanics (CSIR-RMR) system and the RSR system can, with minor modifications be used to classify the rock in the current and future mining areas of the Sydney coalfield. The three systems have each been assessed at a variety of study sites in the Donkin, Prince, Lingan and Phalen Mines. The conclusions reached and the recommendations for further research are presented below:

a. A review of the effects of geological features on coal mining in the Sydney Coalfield indicates that the stability of mine openings are generally controlled by the nature of the rock units and the conditions of contacts between different lithological units rather than by jointing as is the case in most hard rock situations. In fact, the data suggests that thin laminated units and weak contacts between lithological units of markedly different composition tend to control the stability of the excavations.

b. The available rock mass classification schemes, including the three systems under considerations in this study, do not allow direct characterization of the specific weaknesses inherent in coal measures sedimentary rock sequences. In consequence, as part of this study, a need has been recognized for developing classification descriptors which are directly relevant to the rock units present in the coal mining environment of the Sydney Coalfield.

   Appropriate classification descriptions were therefore developed for both the Q and RMR systems. The approach adopted was to establish viable parameter descriptors for each classification component. In order to maintain consistency with the original intents of each of the classifications, the modification process was based on back-analyses of the applied rock loads onto the ring beam support system at Donkin. The data from the installed monitoring points and that from the records of observed performance of the support system was then used for checking the classification approaches. The results of the back-analyses are presented in Table 4.1.

Based on this back-analysis the only classification descriptors that required alteration in the Q system were the input parameter codes for the spacing (J_n) and infilling (J_a) characteristics of discontinuities. Specifically, the J_n and J_a parameters were modified to more closely relate them to describing conditions pertinent to coal measures strata. Similarly, the descriptions for the condition of discontinuities in the RMR system also required modification to describe the contact conditions between different lithological units. These modified descriptor codes are shown in Tables 4.2 and 4.4 respectively for the Q and RMR systems. These two tables therefore form the key references for undertaking future rock mass classifications of the other mining areas of the Sydney Coalfield.

c. In order to facilitate rapid rock mass classification of coal mine roof stability states across the various mines of the Sydney Coalfield, typical rock conditions have been summarized into nine characteristic roof type groupings (A to I) as shown in Figure 5.1 based partly on CBDC categorizations.

Rock mass classifications have been undertaken for selected study areas in the Donkin, Prince, Lingan and Phalen Mines to examine most of these characteristic roof type conditions. The results are presented for the four mines in Tables 4.1, 5.4, 5.6 and 5.8 respectively.

In general, roof types A and B represent good roof conditions. Roof types C to H (roof Type I was not encountered), due to variations in the interbedded units, comprise a range of rock mass quality depending on the type of rock units encountered. In most cases, the classification approaches matched closely the result intuitively expected from appraisal of observed conditions and evaluation of the performance of the installed support. Although it is rather difficult to generalize the rock mass quality purely based on a broad category of rock types, reasonable correlation was achieved by means of the modified classification approaches. However, application of the systems points to the fact that detailed information on the stratigraphy at any location is essential in order to allow a rational evaluation of rock mass quality and hence of required support.

The effect of the advancing longwall on the rock mass characteristics can be taken into account by derating the RMR values and assigning appropriate SRF factors for the Q system for such situations.

d. The correlations between the Q, RMR and RSR values determined from the four mines were examined and the following characteristics of the correlations are noted:

RMR vs Q

The data from the current study can still be reasonably represented by the well known relationship:

$$RMR = 9 \ln Q + 44$$

developed by Bieniawski (1973) from other case records.

RSR vs Q

The following expression, which is identical in form to that for the correlation between the Q and CSIR-RMR systems is reasonably representative of the data collected from the present study:

$$RSR = 9 \ln Q + 44$$

The relationship presented by Rutledge and Preston (1978) also still correlates closely with the collected data.

RMR vs RSR

The correlation of these two parameters seems to fit the expression suggested by Rutledge and Preston (1978).

$$RSR = 0.77\,RMR + 12.4$$

The fact that all of the existing published correlations between the three classifications indices are consistent with the data developed from this study is encouraging for generic application of these systems to coal measure rocks.

e. It is considered that all of the classification schemes examined in this study can be applied within the mines of the Sydney Coalfield. The Q and RMR systems, as modified, however, provide better definition of rock mass conditions than the RSR system as the input parameters for this latter system are not as closely defined. In fact, the RSR values can be developed by the Q and RMR approaches based solely on the correlation between these indices described in item (d) above.

f. It is important to note that the empirical support pressure approaches for the Q and RMR systems were developed on different bases. The support pressure for the Q system varies with the rock mass quality Q and the characteristics and contact conditions of different lithological units. As such, it is independent of excavation geometry. The pressure computation based on the RMR system however is dependent not only on the rock mass quality, it also depends on the span of the opening. In fact, as derived, there is a limiting maximum pressure which is a function of the span only. A similar geometrical term needs incorporating into the Q system as the pressure estimates for very poor quality rock, with this system as currently calculated, far exceed those experienced in practice.

g. For the Donkin Mine where rock conditions are average and thus fall within the central range of both classifications both support pressure estimates are valid. For this situation, the support pressures derived from the Q and RMR systems are in reasonable agreement with those calculated from back-analyses of measured displacements. In other mines, however, there are substantial discrepancies between the empirical support pressure from the classification schemes and those from the back-calculated values. Nevertheless the range of loosened rock back-calculated from the roof beam deflection and the limited extensometer data from Phalen Mine appear to be within the same range estimated from both the Q and RMR systems when both approaches are valid. For poor to very poor rock quality as defined by the Q system, the estimated pressure from the Q values are unrealistic, as no limiting arching action is considered in the calculation.

h. The lack of rock displacement data and support installation records at the Prince, Lingan and Phalen mines makes it difficult to extend the data base derived from Donkin to assess the extent of loosened rock above a given type of installed support. It is considered that if such data can be acquired, the support pressures estimated from the two empirical relationships (given in the Q and RMR systems) could be improved significantly. In the interim, the approaches followed in this study should be tried elsewhere, but the results should be viewed only in relative terms as a general guide for assessing support requirements and not at present as entirely definitive. Further correlations of rock mass classification index data collected from more extensive field work is necessary with physically measured and observed performance of the support. Such correlations would aid considerably in projecting the experience gained from this project to other areas of the mines.

i. In order to assess and refine the classification procedures developed in this study it is recommended that one of the gate roads in the future mining areas in each operating mine be selected for more concentrated study. In each area the following work should be carried out.

- characterize in detail the geological condition of the roof and immediate crown areas of the excavation.
- closely monitor the excavation and support installation procedures.
- monitor the rock displacement of the crown if resources are available or monitor closely the support performance. This may include measuring the deflection of the support frame both in the roof beam and legs.
- classify the rock mass using both the Q and RMR classification systems. [If required, the RSR system can be used but it may be sufficient solely to derive RSR values based on the two other classifications].
- back-calculate the most probable loading on the support using structural analysis on the basis of the support deflections.
- develop correlation charts between these probable loadings and the pressures using the expressions discussed previously.

The above procedure would then provide sufficient site specific quantitative correlation between support pressures and rock mass quality for developing a maximum limiting value expression for the Q index as well as a means of checking the currently available expressions for estimating support pressure. Factors relating to support installation could also be assessed in the light of the support pressure calculations.

j. The ultimate aim of utilizing rock mass classifications in the coal mines may be considered to be in determining support requirements. As the scope of the classifications and the input ratings as modified for the present study have been deliberately cross-checked back to the original intent of the ratings, the published support requirement tables can be used based on the final classification index values calculated from field observations and applications of the methods.

Evaluation of the suitability of the estimated support requirements derived from these classifications, for use in the Sydney Coalfield mines is beyond the scope of this study, however, it should be noted that there are areas in the mines where in our opinion active support systems such as rock bolts could be effectively implemented. The estimates of support requirements given by these classification systems could form the basis for wider future use of this type of support design in the mines of the Sydney Coalfield.

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REFERENCES

Barton, N., Lien, R. and Lunde, J., Engineering Classification of Rock Masses for the Design of Tunnel Support, Rock Mechanics, Vol. 6, No. 4, 1974, pp. 183-236.

Bieniawski, Z.T., Engineering Classification of Jointed Rock Masses. Transactions, South African Institution of Civil Engineers, Vol. 15, No. 12, 1973, pp. 335-344.

Bieniawski, Z.T., Rafia, F. and Newman, D.A. Ground Control Investigations and Assessment of Roof Conditions in Coal Mines. Proc. 21st Symp. on Rock Mechanics, University of Missouri, Rolla, 1980, pp. 691-700.

Bieniawski, Z.T., Rock Mechanics Design in Mining and Tunnelling, A.A. Balkema, Boston, 1984, p. 271.

CANMET (CBCRL) Research Report by Golder Associates Ltd., Measurement and Analysis of Rock, Deformation and Support System response in the Drill and Blast and bored access Drivages at the Donkin-Morien Project (UP-G198), Volume I-IV, June 1985.

CANMET (CBCRL) Research Report by Golder Associates Ltd., Review and Evaluation of Main, Roadway Intersection Techniques for use in the Sydney Coalfield, DSS. File No. 15SQ.23440-7-9010, June 1988.

Deere, D.U., Technical Description of Rock Cores for Engineering Purposes, Rock Mechanics and Engineering Geology, Vol. 1, 1964, pp. 17-22.

Deere, D.U., Hendron,Jr., A.J., Patton, F.D. and Cording, E.J., Design of Surface and Near-Surface Construction in Rock', Symp. Rock Mech., 8th, Minnesota, 1966 (Proc. to be published by IME, 1967).

Deere, D.U. and Miller, R.P., Engineering Classification and Index Properties for Intact Rock, Tech. Report No. AFNL-TR-65-116, Air Force Weapons Laboratory, New Mexico 1966.

Department of Mines and Energy, Coal in Nova Scotia, 1985.

Evans, J.A. and Lawrence, M.S., A Case Study on Past Shortwall Mining Methods and the Risk of Ground Subsidence in the Lanarkshire Coalfield Scotland, Proceedings of the 23rd Annual Conference of the Engineering Group of the Geological Society, Engineering Geology of Underground Movements, September 1987.

Forgeron, S., MacKenzie, B. and MacPherson, K., The Effects of Geological Features on Coal Mining, Sydney Coalfield, Nova Scotia. Cape Breton Development Corporation Report, 1986.

Franklin, J.A., Safety and Economy of Tunnelling, Proceedings, Tenth Canadian Rock Mechanics Symposium, Kingston 1975.

Laubscher, D.H. and Taylor, H.W., The Importance of Geomechanics Classification of Jointed Rock Masses in Mining Operations. Proceedings of the Symposium on Exploration for Rock Engineering, Johannesburg (1976).

Lauffer, H., Gebirgsklassifizierung fur den Stollenbau, Geologie und Bauwesen, Vol. 24, No. 1, 1958.

Priest, S.D. and Hudson, J.A. (1981), Estimation of Discontinuity Spacing and Trace Length using Scanline Surveys. Int. J. Rock Mech. Min. Sci. 18.

Rutledge, J.C. and Preston, R.L., Experience with Engineering Classifications of Rock for the Prediction of Tunnel Support, Proceedings, International Tunnelling Symposium, Tokyo, 1978, pp. A-3-1:7.

Terzaghi, K., Rock Defects and Loads on Tunnel Support. Rock Tunnelling with Steel Supports, eds. R.V. Proctor and T. White, Commercial Shearing Co., Youngstown, Ohio, 1946, pp. 15-99.

Unal, E., Design Guidelines and Roof Control Standards for Coal Mine Roofs, Ph.D. Thesis, The Pennsylvania State University, 1983.

Wickham, G.E., Tiedemann, H.R. and Skinner, E.H., Support Determination Based on Geologic Predictions. Proceedings, Rapid Excavation and Tunnelling Conference, AIME, New York, 1972, pp. 43-64.

Yuen, C.M.K., Boyd, J.M. and Aston, T.R.C., Rock Support Interaction Study of a TBM Driven Tunnel at the Donkin Mine, Nova Scotia. Proc. International Congress on Rock Mechanics, 6, 1339-1344 (1987).

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